9 *Practical Solutions to Mining Problems*

17th *International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4*

Designing a Pullback Dragline Panel for Dipping Coal Seam Conditions

Z.Duran & B.Erdem
Department of Mining Engineering, Cumhuriyet University. Sivas, Turkey

ABSTRACT: This study, in a broad sense, fills in the missing parts in previous efforts at pullback stripping mode design with a dipping coal seam. Three different spoiling procedures have been developed: the normal mode, where the coal seam is flat/nearly flat; the uphill mode, where the coal seam is dipping and the dragline spoils uphill; and the downhill mode, where the coal seam is again dipping but the dragline spoils downhill. The spoiling pattern has a great impact on dragline efficiency. The waste can be spoiled near the set on which the dragline sits or near the set on which the dragline digs Each pattern has been analysed in pit geometry design. The study of pullback design has been extended to cover key cut waste placement in order to operate with various pit width values and spoil-side concerns to control spoil-bound conditions.

1 INTRODUCTION

Pullback stripping is generally applied when the operating dimensions of a dragline are inappropriate for uncovering coal seams without rehandling. The main advantage of this method is that it enables a dragline which has a limited operating radius to handle overburden covers of greater depth than would normally be contemplated. Overburden removal can be performed with a single dragline or a tandem dragline system. When a single dragline is utilized, the dragline takes periodic sojourns across on the spoil pile, getting there either on a section of extended bench or bridge, or around the end of the pit. When a tandem system is used, one machine operates on the highwall side, while the other strips the rehandle material and the barrier left on the highwall side.

Of the previous studies of this topic, that of Cook & Lappİ (1979) can be mentioned İn that it exposed the geometrical interaction between relative dimensions of die dragline and *the* pit with a set of equations for the horseshoe method. Satchwell (1985) studied the pit geometry of the double pass method with rehandle, in which each one of two draglines would be deployed on either side of the pit in an effort to design a dragline pit for the Turkish Coal Enterprises' (TKJ) Elbistan-B open-pit lignite mine. Later, Erdem (1996), Erdem & Çelebi (1998) and Erdem et al. (1999) introduced design guidelines for the pullback stripping method for a flat-lying coal seam on one and two benches, respectively. Duran (2000) improved upon me above-mentioned studies, mainly by incorporating design guidelines for inclined coal seams, different spoiling patterns, key cut excavation and placement procedures. This study presents a pullback model developed for optimal dragline selection.

2 THE PULLBACK MODEL

In pullback stripping, the spoil pile is allowed to ride up the highwall as rehandling is an inherent characteristic of this method. As the dragline is positioned on the set behind the one to be dug, there exist two spoiling pattern alternatives. In the first of these, the waste is dumped into the empty pit near the set on which the dragline is located (Figure 1). This is called the dump-near-sit (DNS) partem. The dimensions used İn the design stage are illustrated in Figures 1-2 and given in the nomenclature. In the second pattern, the waste is dumped into the empty pit near the set which the dragline digs (Figure 3). This is called the dump-near-dig (DND) pattern.

The model comprises of three operating modes (Level, Downhill and Uphill) in each of which two spoiling patterns (DNS and DND) are embedded. In the case of an inclined coal seam, the model analyses downhill and uphill operating principles. In addition, each spoiling partem includes three key cut waste placement procedures (dumping at the toe of the previous spoil pile=>W_{mm}; dumping within pit=>W_m,u; and dumping at the toe of the coal seam^Wmax). Finally, the model can study 18 different operational scenarios for pullback stripping.

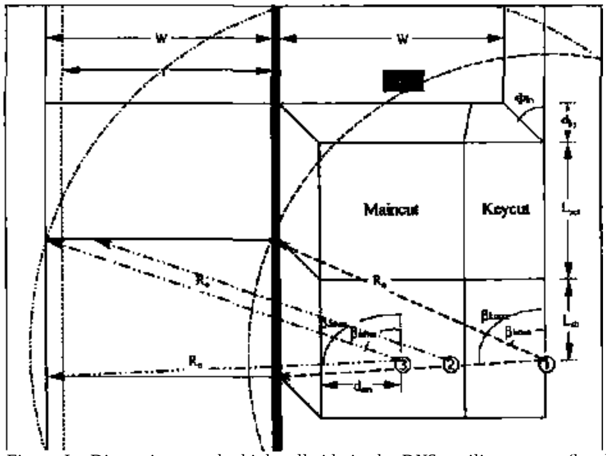


Figure I. Dimensions on the highwall side in the *DNS* spoiling pattern (level coal seam, W^,,).

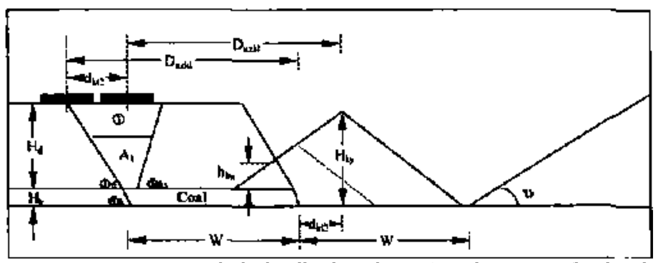


Figure 2. Dimensions on the highwall side in the DNS spoiling pattern (level coal seam, W™,).

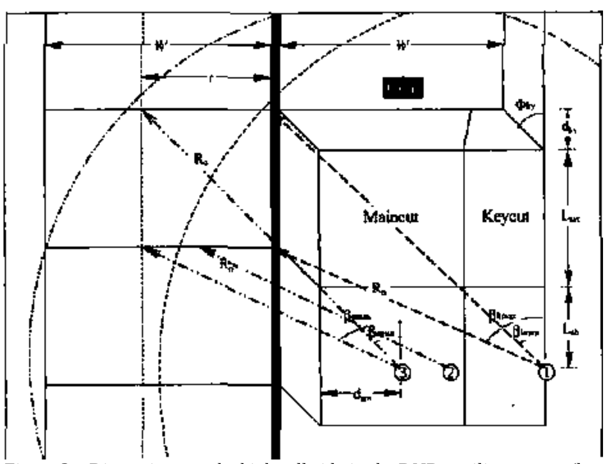


Figure 3. Dimensions on the highwall side in the DND spoiling pattern (level coal seam).

2.1. *Dragline selection*

The pullback model was tested on a virtual strip coal mine with the characteristics given in Table 1. Three draglines, whose main physical characteristics are given in Table 2, were used in the test procedure.

Table 1. Input data for the pullback model.

Overburden thickness	m	30
Coal seam thickness	m	4
Highwall slope angle		65
Coal seam bench angle		65
Angle of repose of waste in spoil pile		38
Swell factor		1.36
Coal seam inclination angle		10

Table 2. Data related to input draglines.

	Operating radius	Tub diameter	Digging depth	Dumping height
	m	m	m	m
#1	77.7	19.4	29.0	38.1
#2	83.8	19.4	33.5	42.7
#3	91.1	19.4	44.2	32.0

The model was executed for all operating modes and spoiling patterns towards reaching common conclusions and rules so that generic operating guidelines for pullback stripping could be formulated. Therefore, me test was conducted for each individual combination of a total of 18 cases. These are given below:

1. Level mode, DNS pattern, W_{min} (Table 3)
2. Level mode, DNS pattern, W_{mjd} (Table 3)
3. Level mode, DNS pattern, W_{max} (Table 3)
4. Level mode, DND pattern, W_{min} (Table 4)
5. Level mode, DND pattern, W_{mid} (Table 4)
6. Level mode, DND pattern, W™« (Table 4)
7. Downhill mode, DNS pattern, W™ (Table 5)
8. Downhill mode, DNS pattern, W_{mid} (Table 5)
9. Downhill mode, DNS pattern, W_{max} (Table 5)
10. Downhill mode, DND pattern, W_{min} (Table 6)
11. Downhill mode, DND pattern, W_{mLd} (Table 6)
12. Downhill mode, DND pattern, W™* (Table 6)
13. Uphill mode, DNS pattern, W_{miD} (Table 7)
14. Uphill mode, DNS pattern, W_{mi}d (Table 7)
15. Uphill mode, DNS pattern, W_{max} (Table 7)
16. Uphill mode, DND pattern, W_{min} (Table 8)
17. Uphill mode, DND pattern, W_{mid} (Table 8)
18. Uphill mode, DND pattern, W_{max} (Table 8)

3 MODEL RESULTS

3.1. *Results of the DNS spoiling pattern*

1. Pit width can be assigned an interval (Wnm^WmHj^Wm,,;,) instead of a single value. However, the downhill mode offers a wider interval than the uphill mode.

2. In level seams and downhill spoiling of inclined seams, the required (Ruzki) and available reach (D_{uz}iti) at the key cut position are independent of pit width and take constant values. In contrast, in uphill spoiling of inclined seams, the required and available reach at the key cut position and the available reach at the main cut position (D^an) are inversely proportional to pit width.

3. Set area (A J is directly proportional to pit width. It increases as the pit width increases and is maximized at the largest pit.

4. In level seams and downhill spoiling of inclined seams, average swing angles at the key cut (ßkey) and main cut positions (ßmain) are directly proportional to pit width. A rise in pit width increases swing angles. Conversely, in uphill spoiling, a rise in pit width decreases swing angles.

Table 3. Results of the piillback model (pattern = dumpi near sit, coal seam = level).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{mlD}	W_{mid}	W^	$W_{m,,,}$	W_{mid}	W «	W_{min}	W*	W™
W	24.25	34.00	44.12	25.92	39.00	51.84	30.34	45.00	60.69
dk.1	19.87	10.12	0.00	0.00	12.84	0.00	0.00	15.69	0.00
Ruzkl	59.97	59.97	59.97	67.69	67.69	67.69	76.54	76.54	76.54
Duikl	59.97	59.97	59.97	67.69	67.69	67.69	76.54	76.54	76.54
H*	2.12	*9.65*	Î7.82	-0.34	7.48	17.86	-1.53	*5.54*	18.09
R.™	88.69	91.13	93.66	89.11	92.38	95.59	90.21	93.88	97.80
D_{uan}	59.97	59.97	59.97	67.69	67.69	67.69	76.54	76.54	76.54
D_y	0.21	3.56	6.78	3.07	6.77	10.28	7.13	10.36	13.96
r	29.57	29.57	29.57	37.29	37.29	37.29	46.14	46.14	46.14
Hba	4.14	6.60	8.95	1.82	4.53	7.10	-0.28	2.09	4.74
sp	2	3	3	2	3	3	2	3	3
dm2	0.00	13.97	10.65	000	16.83	13.18	0.00	20.79	16.81
L«,	25.32	26.28	27.22	30.47	31.82	32.07	36.95	38.23	37.26
A«,	614.04	893.46	1201.00	789.70	1241.10	1662.54	1121.30	1720.53	2261.31
ft*	59.32	59.72	60.11	64.13	64.66	64.76	68.96	69.43	69.08
ft™	59.32	59.72	60.11	64.13	64.66	64.76	68.96	69.43	69.08
Hpp	26.70	27.30	27.74	30.02	30.25	30.37	33.93	33.49	33.13
Wp,	38.47	52.50	66.74	36.08	53.90	71.22	36.84	55 63	75.94
Pr*	33.19	30.49	28.21	26.14	24.71	23.41	16.84	17.68	17.91
R*	57.82	62.88	67.56	58.01	63.05	67.68	59.36	62.93	67.09
L*	103.82	91.28	76.77	120.94	110.41	98.83	138.22	131.74	123.26
Ddh	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00
D*	26.70	27.30	27.74	30.02	30.25	30.37	33 93	33.49	33.13
Pv	97.07	98.31	99.78	93.92	95.96	98.07	91.15	96.26	103.80

Table 4. Results of the pullback model (pattern = dump near dig, coal seam = level).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}
W	-	-	-	24.25	28.00	31.99	24.25	34.00	43.86
d_{kt1}	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00
R_{uzkl}	-	-	-	40.10	43.85	47.84	40.10	49.85	59.71
D_{uzkl}	-	-	-	40.10	43.85	47.84	40.10	49.85	59.71
H_{bk}	-	-	-	15.36	16.51	17.86	15.43	18.09	18.09
R_{uzan}	-	-	-	88.69	89.63	90.62	88.69	91.13	93.59
D_{uzan}	-	-	-	40.10	43.85	47.84	40.10	49.85	59.71
D_y	-	-	-	-3.99	-1.70	0.58	-3.99	1.68	6.65
r	-	-	-	9.70	13.45	17.44	9.70	19.45	29.31
H_{ba}	-	-	-	12.44	11.97	11.36	12.44	11.01	9.01
sp	-	-	-	3	3	3	3	3	3
d_{kt2}	-	-	-	2.86	4.76	7.20	2.96	7.86	10.31
L_{set}	-	-	-	24.18	22.01	19.40	32.40	26.85	19.40
A_{set}	-	-	-	586.37	616.25	620.58	785.64	912.83	850.87
β_{key}	-	-	-	33.83	36.58	39.45	32.59	39.22	45.68
β_{main}	-	-	-	33.83	36.58	39.45	32.59	39.22	45.68
H_{pp}	-	-	-	19.67	21.38	23.09	19.67	23.91	27.65
W_{pp}	-	-	-	52.96	55.89	58.81	52.96	60.21	66.58
P_{reh}	-	-	-	40.80	38.42	35.88	40.80	34.60	28.38
R_g	-	-	-	63.32	64.68	65.80	63.32	66.25	67.54
L_{ys}	-	-	-	109.79	106.55	103.78	131.00	125.06	122.27
D_{dh}	-	-	-	30.00	30.00	30.00	30.00	30.00	30.00
D_{ds}	-	-	-	19.67	21.38	23.09	19.67	23.91	27.65
β_y	-	-	-	102.80	101.92	101.09	101.76	99.84	98.36

Table 5. Results of the pullback model (pattern = dump near sit, coal seam = inclined, downhill spoiling).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}
W_y	24.25	31.63	39.65	24.25	36.15	47.70	27.20	42.48	56.84
d_{kt1}	15.40	8.02	0.00	23.45	11.55	0.00	0.00	14.36	0.00
R_{uzkl}	57.20	57.20	57.20	65.24	65.24	65.24	74.39	74.39	74.39
D_{uzkl}	57.20	57.20	57.20	65.24	65.24	65.24	74.39	74.39	74.39
H_{bk}	1.32	6.34	11.77	-2.86	3.06	10.97	-5.23	0.46	10.22
R_{uzan}	77.43	78.07	78.75	77.43	78.45	79.44	77.69	79.00	80.23
D_{uzan}	57.20	57.20	57.20	65.24	65.24	65.24	74.39	74.39	74.39
D_y	-7.24	-5.28	-3.41	-5.81	-3.62	-1.71	-3.21	-1.81	-0.38
r	25.10	25.10	25.10	33.15	33.15	33.15	42.29	42.29	42.29
H_{ba}	3.18	4.50	5.75	-0.09	1.39	2.67	-3.15	-2.20	-1.24
sp	2	3	3	2	3	3	2	3	3
d_{kt2}	0.00	10.59	7.29	0.00	13.99	10.32	0.00	18.35	14.82
L_{set}	20.94	21.61	22.19	25.59	26.44	27.11	31.55	32.15	32.72
A_{set}	507.91	683.64	879.89	620.55	955.77	1293.13	858.37	1365.61	1859.88
β_{key}	54.22	54.48	54.70	59.32	59.64	59.90	64.47	64.69	64.89
β_{main}	54.22	54.48	54.70	59.32	59.64	59.90	64.47	64.69	64.89
H_{pp}	31.89	33.07	34.11	35.13	36.15	36.92	39.46	39.47	39.56
W_{pp}	46.16	58.53	71.63	39.94	58.65	76.51	38.07	60.27	81.28
P_{reh}	23.82	20.73	17.91	16.21	13.91	11.98	4.12	4.86	4.96
R_g	48.89	50.35	51.41	47.20	47.62	47.58	46.25	43.96	42.02
L_{ys}	120.78	118.36	116.52	138.49	137.90	137.96	156.97	159.58	161.66
D_{dh}	33.19	33.19	33.19	33.19	33.19	33.19	33.19	33.19	33.19
D_{ds}	21.36	20.90	20.21	24.00	22.55	20.96	26.97	24.04	21.36
β_y	92.55	96.33	103.71	92.26	102.08	112.48	98.38	110.33	123.00

Table 6. Results of the pullback model (pattern = dump near dig, coal seam = inclined, downhill spoiling).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}
W_y	-	-	-	24.25	25.00	25.35	24.25	31.00	38.28
d_{kt1}	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00
R_{uzkl}	-	-	-	41.79	42.54	42.89	41.79	48.54	55.82
D_{uzkl}	-	-	-	41.79	42.54	42.89	41.79	48.54	55.82
H_{bk}	-	-	-	11.54	11.78	11.90	11.76	12.92	12.16
R_{uzan}	-	-	-	77.43	77.50	77.53	77.43	78.01	78.64
D_{uzan}	-	-	-	41.79	42.54	42.89	41.79	48.54	55.82
D_y	-	-	-	-8.47	-8.16	-8.01	-8.47	-5.91	-3.76
r	-	-	-	9.70	10.45	10.80	9.70	16.45	23.73
H_{ba}	-	-	-	10.45	10.26	10.18	10.45	8.62	6.24
Sp	-	-	-	3	3	3	3	3	3
d_{kt2}	-	-	-	1.89	2.38	2.62	2.22	5.51	7.16
L_{set}	-	-	-	20.04	19.61	19.40	28.36	24.50	19.40
A_{set}	-	-	-	486.06	490.17	491.78	687.65	759.46	742.73
β_{key}	-	-	-	34.19	34.74	34.99	32.89	37.45	42.25
β_{main}	-	-	-	34.19	34.74	34.99	32.89	37.45	42.25
H_{pp}	-	-	-	27.07	27.48	27.66	27.07	30.47	33.58
W_{pp}	-	-	-	59.99	60.64	60.95	59.99	65.53	70.73
P_{reh}	-	-	-	29.12	28.58	28.32	29.12	24.21	18.91
R_g	-	-	-	55.13	55.09	55.07	55.13	54.19	51.95
L_{ys}	-	-	-	126.22	126.29	126.33	145.05	146.46	149.66
D_{dh}	-	-	-	33.19	33.19	33.19	33.19	33.19	33.19
D_{ds}	-	-	-	17.43	17.61	17.70	17.43	18.90	20.04
β_Y	-	-	-	95.81	95.69	95.64	95.35	94.52	100.25

Table 7. Results of the pullback model (pattern = dump near sit, coal seam = inclined, uphill spoiling).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}
W_y	24.25	28.78	33.04	26.48	34.10	41.57	30.93	41.57	53.29
d_{kt1}	19 06	13.87	8.97	0.00	15.84	7.37	0.00	16.69	3.51
R_{uzkl}	58.18	57.52	56.88	65.82	64.83	63.81	74.39	73.13	71.68
D_{uzkl}	58.18	57.52	56.88	65.82	64.83	63.81	74.39	73.13	71.68
H_{bk}	6.12	10.56	15.73	4.13	11.26	18.58	3.69	11.90	23.55
R_{uzan}	109.58	111.76	113.81	110.65	114.33	117.92	112.80	117.92	123.56
D_{uzan}	58.18	57.52	56.88	65.82	64.83	63.81	74.39	73.13	71.68
D_y	4.66	6.41	8.02	8.54	11.16	13.66	13.67	16.85	20.41
r	28.76	28.09	27.46	36.40	35.40	34.38	44.96	43.71	42.25
H_{ba}	7.12	8.92	10.59	5.46	8.15	10.77	4.22	7.52	11.25
sp	2	3	3	2	3	3	2	3	3
d_{kt2}	0.00	16.49	16.36	0.00	20.97	18.03	0.00	23.78	19.22
L_{set}	23.94	23.12	22.34	29.80	28.42	27.07	36.39	34.48	32.36
A_{set}	580.65	665.34	738.25	788.98	969.19	1124.98	1125.69	1433.14	1724.45
β_{key}	56.57	55.45	54.40	61.62	59.91	58.23	66.23	64.11	61.73
β_{main}	56.57	55.45	54.40	61.62	59.91	58.23	66.23	64.11	61.73
H_{pp}	26.14	27.08	27.93	28.83	30.14	31.38	32.28	33.72	35.36
W_{pp}	35.39	41.54	47.27	34.31	44.11	53.67	35.50	48.43	62.83
P_{reh}	47.91	46.92	46.07	42.97	42.35	41.75	36.80	37.27	37.43
R_g	69.08	73.55	77.63	70.86	77.38	83.65	74.56	82.27	90.99
L_{ys}	71.15	50.13	6.47	89.46	64.36	10.07	104.69	78.24	9.13
D_{dh}	32.10	32.84	33.53	32.46	33.71	34.92	33.19	34.92	36.83
D_{ds}	29.65	30.20	30.68	33.74	34.44	35.08	38.64	39.29	40.04
β_Y	110.85	112.02	113.16	106.28	108.27	110.28	102.27	105.04	108.12

Table 8. Results of the pullback model (pattern = dump near dig, coal seam = inclined, uphill spoiling).

Variable	Dragline #1			Dragline #2			Dragline #3		
	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}	W_{min}	W_{mid}	W_{max}
W_y	-	-	-	24.25	26.00	28.64	24.25	31.00	39.18
d_{kt1}	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00
R_{uzkt}	-	-	-	39.13	40.88	43.52	39.13	45.88	54.06
D_{uzkt}	-	-	-	39.13	40.88	43.52	39.13	45.88	54.06
H_{bk}	-	-	-	19.37	20.02	21.26	19.43	22.51	24.99
R_{uzan}	-	-	-	109.58	110.42	111.69	109.58	112.83	116.77
D_{uzan}	-	-	-	39.13	40.88	43.52	39.13	45.88	54.06
D_y	-	-	-	-1.49	-0.10	1.94	-1.49	3.73	9.69
r	-	-	-	9.70	11.45	14.09	9.70	16.45	24.63
H_{ba}	-	-	-	14.05	14.06	14.05	14.05	14.01	13.67
sp	-	-	-	3	3	3	3	3	3
d_{kt2}	-	-	-	6.70	7.41	8.90	6.79	10.53	13.87
L_{set}	-	-	-	22.61	21.37	19.40	30.77	26.11	19.40
A_{set}	-	-	-	548.19	555.62	555.64	746.19	809.31	759.98
β_{key}	-	-	-	32.53	33.74	35.55	31.33	35.67	40.73
β_{main}	-	-	-	32.53	33.74	35.55	31.33	35.67	40.73
H_{pp}	-	-	-	21.66	22.54	23.84	21.66	24.97	28.68
W_{pp}	-	-	-	46.57	47.85	49.72	46.57	51.34	56.64
P_{reh}	-	-	-	53.00	52.12	50.77	53.00	49.57	45.47
R_g	-	-	-	72.85	74.34	76.47	72.85	78.27	83.85
L_{ys}	-	-	-	82.82	77.34	68.54	109.39	93.24	71.21
D_{dh}	-	-	-	32.10	32.38	32.81	32.10	33.20	34.53
D_{ds}	-	-	-	21.31	22.45	24.13	21.31	25.61	30.53
β_y	-	-	-	117.22	116.75	116.07	114.88	113.33	111.72

5. In level seams and downhill spoiling of inclined seams, the effective reach (r) is independent of pit width and takes constant values. In contrast, in uphill spoiling of inclined seams, it is inversely proportional to pit width.

6. In level seams and downhill spoiling of inclined seams, set length (L_{Kt}) is directly proportional to pit width. However, marginal increase in the set length is less than that in the pit width. In uphill spoiling of inclined seams, set length is inversely proportional to pit width.

7. The rehandle percentage ($P_{re}h$) decreases as pit width increases. It is lower in downhill mode than in uphill mode for the same operating conditions.

8. The height the key cut spoil (Hbk) and the main cut spoil (Hba) climb on the highwall increases with the pit width. Here, marginal increase in Hbk is more than that in Hba- This indicates that die wider the pit is, the larger the part of the coal seam that is buried under the waste barrier.

9. As the pit gets wider, the number of points on which the dragline is positioned (sp) increases from 2 to 3 due mainly to excavation of the key cut from 2 points.

10. As the pit gets wider, the required reach on the spoil side (Rg) increases, but the set length on the spoil side (Lys) decreases. In addition, R_g is longer in uphill mode than downhill mode for the same operating condi tions. Therefore, draglines with limited reach may easily fail to operate on the spoil side in uphill mode.

11. The swing angle on the spoil side (β_y) is directly proportional to pit width.

12. In uphill spoiling of inclined seams, the required reach at the main cut position is greater than that in downhill mode.

13. Draglines with limited dumping height capability fail to operate in uphill spoiling mode.

3.2. *Results of the DND spoiling pattern*

1. The required and available reach at the key cut position and the available reach at the main cut position increase with pit width. The required reach at the main cut position is directly proportional to pit width.

2. The set area becomes greater to a certain level of pit width and then shrinks with greater pit width values. The cause of this is that the marginal decrease in set length corresponding to marginal increase in pit width İs less up to a certain value of pit width, and, consequently, set area gets larger. After a certain width, the situation is reversed and the set area shrinks.

3. Average swing angles at the key cut and main cut positions are directiy proportional to pit width.

4. The effective reach is directly proportional to pit width.

5. The height the main cut spoil climbs on the highwall decreases as pit width increases. The height the key cut spoil climbs on the highwall maintains an almost constant level except for narrow pits on flat seams.

6. The rehandle percentage ($P_{re}h$) decreases as pit width increases. It is lower in downhill mode than in uphill mode for the same operating conditions.

7. In some cases where the key cut cannot be formed, the dragline excavates the whole pit from on the main cut position.

8. Draglines with limited dumping height capability fail to operate in uphill spoiling mode in narrow pits.

9. The required reach on the spoil side is greater in uphill mode than in downhill mode for the same operating conditions. Therefore, draglines with limited reach may easily fail to operate on the spoil side in uphill mode.

3.3. *Results of the pullback model on the basis of spoiling patterns*

1. The swing angles at me key cut and main cut positions in the DND mode are smaller man those in the DNS mode for the same operating conditions.

2. For a certain waste thickness, the DND mode imposes tighter constraints on draglines. Therefore, it is very likely that a dragline that has failed in the DND mode can operate with the DNS mode.

3. In DNS mode, a wider interval of pit width values is offered than in DND mode for the same operating conditions.

4. In DNS mode, greater reach from the key cut and main cut positions are available than in DND mode for the same operating conditions. Hence, the effective reach is also greater.

5. In DND mode, the rehandle percentage is greater than in DNS mode for the same operating conditions. The heights the key cut spoil and the main cut spoil climb on the highwall are greater.

6. In DNS mode, the effective reach is greater than in DND mode for the same operating conditions.

4 CONCLUSIONS

In this study a pullback model was developed for dragline stripping. The model is able to analyse situations in which a dragline operates on a bench that overlies one flat-lying or inclined coal seam.

The model comprises three operating modes (Level, Downhill and Uphill), İn each of which two spoiling patterns (DNS and DND) are embedded. Each spoiling pattern includes three key cut waste placement procedures (dumping at the *toe* of the previous spoil pile=*W_{min}; dumping within pit=>$W_m$id; and dumping at the toe of the coal seam=>W_{max}). Finally, the model can study 18 different operational scenarios for pullback stripping.

The main conclusions drawn from the study are as follows:

1. Draglines are allowed to dig thicker waste in downhill mode. For this reason, small-sized draglines may fail to operate in uphill mode.

2. The uphill mode requires that the dragline have a greater dumping height and the dragline cannot make good use of the available spoil room. Thus, the pit must be kept shorter in this operating mode than in me downhill mode.

3. In the case of an inclined coal seam, the downhill mode should be utilized. However, should stability be of concern, then the uphill mode can be applied as it offers a relatively safe operating environment. Another particular feature is the rehandle percentage. Uphill spoiling always produces higher percentages.

4. The DNS spoiling pattern is preferred. Although swing angles in this pattern are higher than mose in the DND pattern, they are more than compensated for, mainly by lower rehandle percentages and other dimensional benefits.

REFERENCES

Cook, *F.* & Lappi, R.L. 1979. *Operating guidelines far dragline stripping systems' analysis of tandem systems,* Washington: U.S. Department of Energy, Division of Fossil Fuel Extraction, 468 p.

Duran, Z. 2000. *Tek kömür damarlı dragline panolarının üç boyutlu tasarımı* (In Turkish), M.Sc. thesis, Sivas: Cumhuriyet Üniversitesi Fen Bilimleri Enstitüsü, 172 p.

Erdem, B. 1996. *Development of an expert system for dragline and stripping method selection in surface coal mines,* Ph.D. thesis, Ankara: Middle East Technical University School of Natural and Applied Sciences, 383 p.

Erdem, B. & Çelebi, N. 1998. An approach to design a dragline pit in pullback stripping method, *Int. J. of Surface Mining, Reclamation and Environment,* 12:189-195.

Erdem, B., Çelebi, N. & Paşamehmetoğlu, A.G. 1999. A methodology for allocating overburden to slices for dragline stripping, *Proc. 8th Int. Symp. on Mine Planning and Equipment Selection,* Ukraine.

Satchwell, P.C. 1985. *Design of a dragline pit for Turkish Coal Enterprise's (TKİ) Elbistan-B open pit lignite mine* (Unpublished), 25p.

NOMENCLATURE

β_{key}	Swing angle from key cut position (°)
β_{main}	Swing angle from main cut position (°)
β_{y}	Swing angle on spoil side (^{Q})
Φ_{d}	Highwall angle (°)
Φ_{k}	Coal seam bench angle (°)
Φ_{ky}	Cut face angle (°)
θ_{y}	Angle of repose of waste in the spoil pile(°)
A_{set}	Area of set to be dug (m^3)
d_{em}	Safety margin from the highwall (m)
d_{ky}	Cut face distance (m)
D_{dh}	Required digging depth on highwall side (m)
D_{ds}	Required digging depth on spoil side (m)
d_{k11}	Distance dragline reaches from on key cut position (m)
d_{k12}	Distance between key cut excavation positions (m)
D_{uman}	Available operating radius at main cut position (m)
D_{uakl}	Available operating radius at key cut position (m)
D_{y}	Required dumping height on highwall side (m)
H_{bs}	Height to which main cut spoil climbs on highwall (m)
H_{bk}	Height to which key cut spoil climbs on highwall (m)
H_{ky}	Height of key cut spoil inthepit(m)
H_{pp}	Height of pullback pad (m)
L_{sb}	Setback distance (m)
L_{set}	Set length on highwall side (m)
L_{ys}	Set length on spoil side (m)
P_{reh}	Rehandle percentage (%)
r	Effective reach as measured from coal seam toe (m)
R_{g}	Required reach on spoil side (m)
R_{o}	Operating radius (m)
R_{uman}	Required operating radius ai main cut position (m)
R_{uakl}	Required operating radius at key cut position (m)
sp	Number of excavation positions on highwall side
W	Pit width (m)
W_{pp}	Width of puUback pad (m)
W_{y}	Pit width in the case of inclined coal seam (m)

17th International Mining Congress and Exhibition of Turkey-IMCET 2001, ©2001, ISBN 975-395-417-4

Visual Aspect of Mine Evaluation; A Case Study

K. Erarslan
Dumlupinar University, Department of Mining Engineering, Kütahya, Turkey

ABSTRACT: Drill hole data allows for evaluation of the mine area under investigation. The topographical, stratigraphical and assay information are utilized to estimate the dimensions and reserve of the ore body. In this study, a 3 dimensional evaluation system for the topography and ore body has been developed. Input to the system is x (East), y (North) and z (elevation) coordinates of drill holes coupled with stratigraphical and assay values. The system provides modeling of the topographical surface and ore body blocks. A gridded surface is generated to represent the topography. Nodes on the grid are assigned *(x.y.z)* coordinates. Either inverse distance square method, geostatistical methods or neural network system can be used to interpolate the drill hole data through the area. Underground modeling is performed by interpolating available data to three-dimensional blocks, which represent the ore body. According to cut-off grade, blocks are classified as ore or waste. *(x,y,z)* coordinates and assay values of each block are stated in a special and familiar visualization technique, called as DXF format. The system provides the DXF formatted appearance of both topography and ore body as well as 3D drill hole distribution. Photograph-quality visualization js realized by processing in any DXF processing system like AutoCAD, Bryce and 3D Studio Max The reserve is also calculated. The system is coded in C++ and applied in the Çöllolar coal district. The results have revealed the validity of the system.

1 INTRODUCTION

Valuation is a crucial step of mining. The exploration data is evaluated and worth of site under investigation is determined. Reserve, shape, quality distribution and thickness are also determined at this stage. This information is valuable for future analyses. Economical aspect of the work is crucial as well as technical aspect.

In mine evaluation work, the topography, location of drill holes, ore body itself are required to be visualized in 2 dimensions and 3 dimensions. A successful and precise viewing enables seeing the invisible ore body. For this purpose, visualization takes an important role and Is an urgent aspect of evaluation. Since the introduction of computers, this fact has not changed. The computers have merely provided greater ease, speed and accuracy in visual evaluation.

Many evaluation techniques have been developed over time. The numerical results are also supported by visuals. Both academic (Roninson et.al-, 1998; Honerkamp and Mann, 1996) and commercial programs have been coded for a practical and accurate assessment (AutoDESK, 1987; Eagles, 1992; Gemcorn, 1992; Geostat, 1992; Medsystem, 1994; Sur-' pac, 1998). The commercial software allow for comprehensive applications. However, in some deposits having special geological features that software do not consider, they might not be used. In this study, a drill-hole based evaluation system has been developed. Any programmer engineer can develop own evaluation system by means of the approach employed in this study. The system is capable of 3D drill hole diagramming, 3D topographical and ore body modeling, and reserve estimation. The system is applied to a restricted part of the Çöllolar coal district in Elbistan, Turkey. In comparison to previous studies (Erarslan, 1991), reasonable results have been obtained, revealing the applicability and validity of the system.

2 SCOPE OF THE SYSTEM

The visual evaluation system is capable of determining the topographical structure, drill hole locations and ore body extensions in three dimension.

The drill hole *(x.y.z)* coordinates represent Easting, Northing and Elevation. They are used to form a uniform data structure on a superimposed grid system. Data interpolation is provided by inverse distance methods, geostatistics, or neural networks. The *(x,y)* coordinates on the grid can simply be calculated. However, for other parameters like the z coordinate, thickness, grade, etc., the special mathematical methods mentioned above are utilized. Thus, in general, interpolation is made for parameters other than Easting and Northing. The elevation is stated to be visualized.

In addition, the block model of the deposit is also a type of data interpolation performed for the ore blocks. The bench level data from drill holes are interpolated to 3-dimensional blocks. This interpolation is made either in the center of blocks or the corner points.

The visualization of drill holes is another capability of the system. The *(x,y,z)* coordinates of drill holes and their depths are used for modeling.

The developed system utilizes the output of all the extension methods mentioned above in a definite format The system can produce the data eXchange Files *(DXF)* format to describe the *(x.y.z)* coordinates of assigned grid points, drill holes and ore blocks. This is a special and familiar drawing format, which can be processed in several drafting packages like AutoCAD, 3D Studio Max and Bryce. Any mining engineering researcher can develop their own visualization system according to their needs. This is the major advantage of the DXF approach. Once the view is transferred to the drafting packages, any service provided by them can be used for the models.

3 APPLICATION OF THE SYSTEM

3.1. Çölloîar Coal District

The developed *system* has been applied in *the* Çellolar coal district, Elbistan, Turkey. The basin is the largest lignite region in Turkey with more than 3,500,000,000 tons of coal. The average calorific value is between 1000 and 1100 kCal/kg. The stripping ratio is 2.5:1 - 3.0:1. The coal seam İs irregular and tilts up slightly to the north. Drill hole exploration takes place in an area of 5.5 km (east-west) by *8.5 km.* (north-south)

3.2 Three Dimensional Visual Evaluation

The system is introduced using 239 drill holes from the ÇÖllolar district. Their *(x.y.z)* coordinates and depth information are used to represent their distribution in a 3-dimensional aspect. The location map of the holes is shown in Figure 1, while in Figure 2 the three-dimensional view is represented. As can be seen on the Figure 1 and Figure 2, the drill holes are distributed homogeneously, which provides a good and reliable base for further calculations. Figure 3 represents the grid structure superimposed on the district. The resolution of the mesh wire is 40x40 and the radius of influence is estimated to be 500 m in geostatistical studies. Figure 4 shows an isometric view of the drill holes and topography. The figures feature some exaggeration in the z coordinate since the displacement in the east-west and north-south directions is much more greater than the elevation difference.

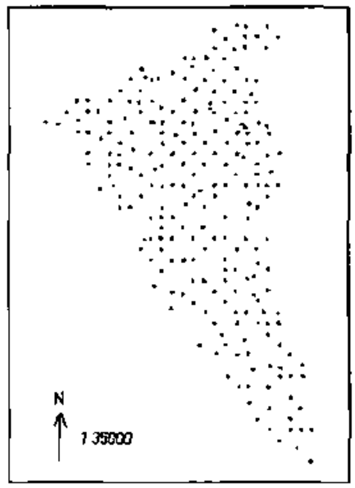


Figure I Location map of the drill holes.

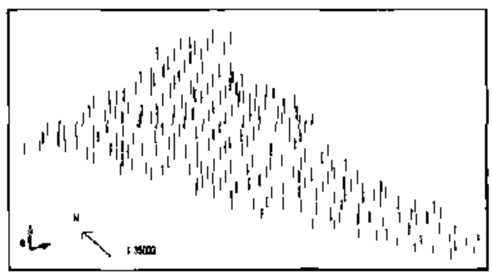


Figure 2 Three-dimensional distribution of drill holes.

Figure 3 Grid mesh imposed on the district.

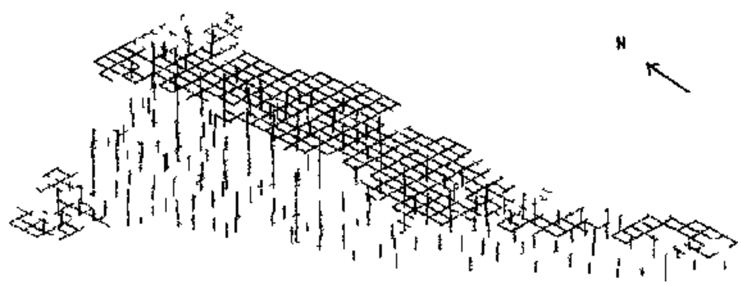


Figure 4 Dnll holes and topography

FigureS Shaded view of topography

The system enables vısuali2atıon of the topography The visual appearance is based on the grid nodes, whose values are assigned by data mterpolation methods The *(x,y,z)* coordinates are calculated and stated in a special drawing format The DXF formatted view of the topography is processed by the Bryce package and after rendenng photograpic quahty views are obtained Figure 6 is an example of rendered topography A 3 dimensional frame is rendered by earth matenal Figure 7 also shows another rendenng matenal from another perspective The visualization has photograpic quality when other effects like clouds are added

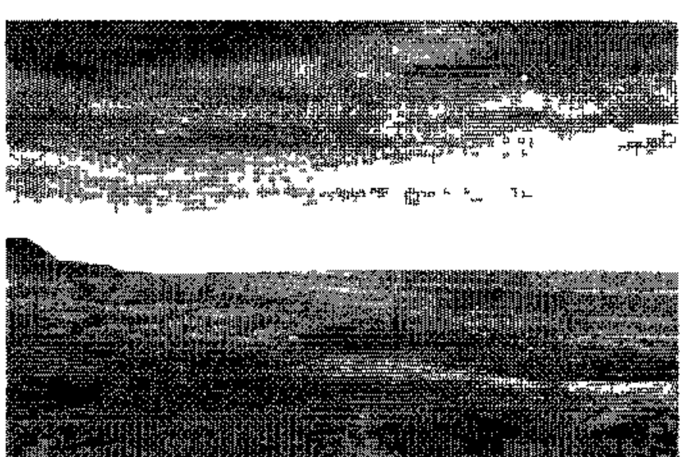


Figure 6 A zoomed view of topography rendered with earth material (some exaggeration in z direction)

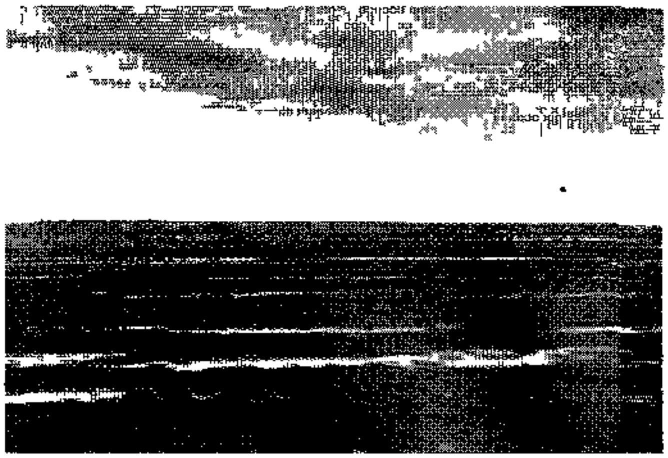


Figure 7 Another zoomed view from the topography with a different rendering materia!

The system interpolates assay values level by level to assign grade or calorific value to blocks A cutoff grade or calorific value will yield the view of the valuable part of the ore body Figure 8 represents 3-dımensıonal block model of the distnct

As can be seen in the figure, the coal seam has a slight tilt (exaggeration in the figure) toward the north The topography and the coal deport can also be visualized together (Fig 9)

The reserve of the ore body, estimated by evaluating 239 drill holes and the ore body block model is 958 3 million tons In comparison to previous studies (Erarslan, 1991), reasonable and logical results reveal that the system developed is suitable and valid for all DXF format processors

Figure 8 Block model of coal seam after rendering

Figure 9 Isometric rendered view of topography and underlying coal seam

The system has also been tested for non-coal deposits (Erarslan, 2000). It has revealed its applicability to all deposits.

4 CONCLUSIONS

Topographical views, 3D ore body models and drill hole diagrams are the visual results of mine evaluation. In this study, a visual evaluation system has been developed. The system is based on drill hole data. Three-dimensional drill hole distribution is visualized by the system. The topography and ore body block model can also be represented İn three-dimensional space. This provides to observe the extensions of the ore body and its underground position. The system also allows for calculation of the reserve using a block model. The outputs of inverse distance methods, geostatistics and neural network systems can be utilized İn visualization. The interpolated data is represented in Data exchange File (DXF) format, which is a simple and practical method of visualization. The system has been coded in C++ and successfully applied in the Çöllolar brown coal district in Elbistan, Turkey.

REFERENCES

AutoDESK, Inc., 1987. The AutoCAD Drafting Package, *Installation and Performance Guide,* 426 S.

Eagles, 1992. Morrison and Knudsen Co., Inc.. Idaho, USA.

Erarslan, K., 1991. "Development of a Coal Deposit Evaluation System", MSc. Thesis. METU, Ankara, 117 p.

Erarslan, K., 1999. "3D Modeling of Deposits: ETİ Copper Enterprise Kütahya-Gümüşkoy Application". *J. DPI! Natural andAppiScL,* Vol. 1, No1, p. 135-148

Gemcom, 1992. Gemcom Services Inc., Vancouver, Canada.

Geostat, 1992. Geostat International, Inc., Montreal, Canada.

Honerkamp, M. and Mann, C, 1998. "Application of Computer Based 3D Visualization in Quarry Planning", *26"' APCOM '96,* 16-20 Sept., Ed. Ramani, R.V., Colorado, pp. 291-294.

Medsystem, 1994. Mintech, Inc., Tucson, Arizona, USA.

Robinson, A.J., Gibbs, A.D., Grifiths, P.A.. Murray, T.,"Application of Structural Modelling and Validation in the Underground Mining Industry" *27"' APCOM '98,* 19-23 April, IMM, London, pp. 59-68.

Surpac, 1998. Surpac Inc., Nothingam, UK.

17th International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

Draw Control Optimisation in the Context of Production Scheduling

M.L.Smith & D.Rahal
Department of Mining, Metallurgical and Materials Engineering/JKMRC, The University of Queensland, Brisbane, Australia

ABSTRACT: Draw control in caving operations involves a combination of scheduling and geomechanics. Geomechanical issues related to draw control have played the dominate role in past efforts to reduce stress, improve fragmentation and reduce dilution. Production scheduling algorithms have been more commonly applied in surface mining but can be used to integrate more traditional methods of draw control with production and development schedules for the life of a panel with the objective of maximising project value. The state-of-the-art in production schedule optimisation is reviewed and compared against the complications related to caving. A generalised outline of the draw control optimisation approach being pursued at the JKMRC is presented.

1 INTRODUCTION

Block and panel caving operations are conducted in massive, generally low grade, deposits, having both weak ore and waste which will readily fragment and flow once undercut (Figure 1). De Beers' Premier and Kofflefontein are illustrative of these methods

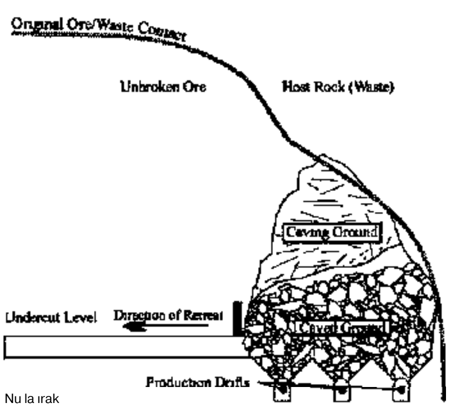


Figure 1 Pane! cave extraction system.

At the Premier mine undercut levels are situated 15 m above their production levels. Caving normally starts once the undercut dimensions reach 100 x 100 m. Advanced undercutting is used at Premier, such that the extraction drawbells are developed after the undercut has passed over. The lag time between undercutting and completion of the extraction level can be no more than 6 months or there is the risk of ore compaction. The point loads developed when ore compaction occurs can damage the production level and necessitate rehabilitation of the mine workings before production can begin. In extreme cases, whole production drifts can be put out of service for a considerable length of time. The development pattern used at Premier is an offset herringbone. Parallel drifts are driven through the orebody and angled crosscuts connect the drifts at 15m intervals (the drawpoint separation distance has been increased to 18 m in some areas of the mine). The drawpoints on opposite sides of the tunnel are offset so that the drawpoints are all at different distances along the drift. The angled entry into the drawpoint crosscuts facilitates LHD entry. After cross cut development, a raise bore is driven upwards to the undercut level to provide a free face for blasting of the draw trough.

There needs to be tight coordination between development of the undercut and the maturation and activation of drawpoint production. If production lags behind development, stress will build on the production level instead of transferring to the boundaries of the panel. This will result İn compaction of the ore, increased incidence of hangups in drawpoints and, in extreme cases, loss of tunnels and drawpoints in the production level.

At Koffiefontein Mine, the Front Caving method is used. This is a combination of block caving and sublevel caving that extracts ore on two production

Levels using a series of semi permanent drawpoints (SPD). A cross section of this layout can be seen in Figure *2.* This method offers two cost related advantages when compared to traditional caving. The first is the low cost of drawpoint support when mining in retreat, consequently the SPDs do not have to last the life of the mine. Undercutting costs are deferred by undercutting simultaneously the two production levels. Production is split between the two levels with an uneven production split between levels 48 and 49 (40 and 60 % of production respectively). Ore draw is also limited, initially, to a maximum daily draw down of 400 mm/day.

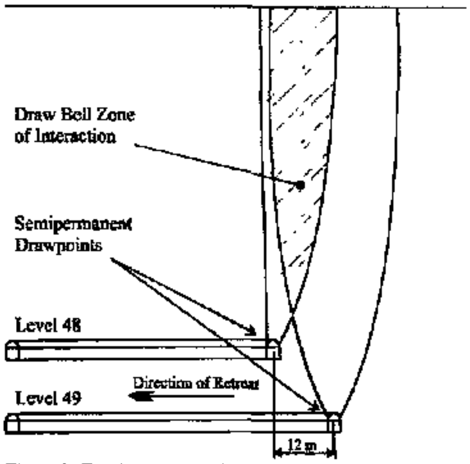


Figure 2. Fronl cave extraction system.

In sublevel caving, the orebody is blasted whilst the surrounding waste rock collapses during draw. Operations in the orebody are undertaken in roadways developed at relatively small vertical intervals. Scheduling of development in advance of production becomes extremely complex with place-changing of jumbos, LHDs and production drills occurring on the roadways, sublevels and production level. Ore is fragmented using blast holes drilled upwards in fans from these headings, allowing the waste rock to cave, then ore is extracted by front end loaders from the production drifts (Figure 3). As broken ore is extracted at the drawpoint, fragmented ore and enclosing caved waste *displace* to fill *the* void. Brady and Brown (1993) note that the mining method is characterised by relatively high dilution and low ore recovery, which has limited its application. Nevertheless, there is currently a resurgence of interest in sublevel caving as an underground mass mining method İn Australia due partially to the increasing cost of supported methods of mining, particularly those requiring backfill.

All these methods of caving share some key characteristics:

- Close sequencing *of* development, undercutting and production
- Steady production to maintain fragmentation and flow and reduce stress
- Mobility and mixing of ore as a function of production
- Frequent and largely unpredictable loss of drawpoints
- Production sourced from a single block for a substantial portion of the mine's life
- An emphasis on minimising dilution throughout a panel's life

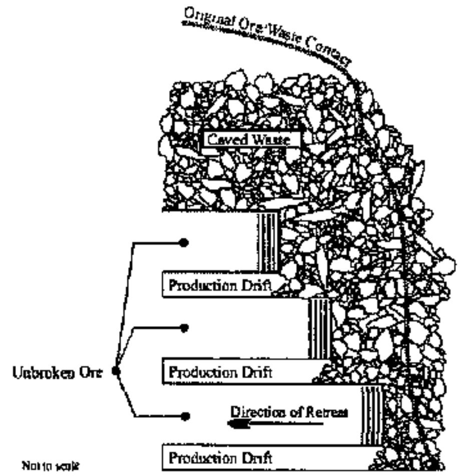


Figure 3. Sublevel cave extraction system.

In the larger context of life-of-mine profitability, draw control must be viewed as the central strategy mat integrates all of these characteristics of caving methods. Draw control encompasses sequencing and scheduling of development, production and the materials handling system with the dual objectives of minimising mining costs and dilution. Thus, draw control cannot be limited to the individual drawpoint but must account for the scheduling of drawpoint production over the life of the panel. Failure to closely follow undercut progression with activation of the underlying drawpoints will result in compaction of the fragmented ore, poor fragmentation and propagation of the cave and transferral of stress to the production level.

Mixing of the fragmented ore occurs both vertically and horizontally. Vertical mixing is a function of the fragment size distribution and results from fines rilling though coarser material. In the case of

Controlling blast fragmentation. In block caving, horizontal mixing results from uneven draw with ore migrating to columns of more mature (heavier drawn) draw points. As a result of mixing between columns, the reserve model is dynamic rather than static as is the case when scheduling is based on block models associated with surface mines or other smaller-scale stoping methods. Therefore, it is extremely difficult to solve the block cave scheduling problem simultaneously for multiple production periods since the contents of the columns in the reserve model are a function of production. The effectiveness of time-dynamic optimisation becomes even more questionable when the availability of the drawpoints and production tunnels is considered as these are frequently out of action due to hangups and failure.

The need for a rational approach to draw control is particularly important in terms of the longevity of a panel. One panel is likely to be the main source of production for many years. Poor draw control early in the panel's life can result in compounding difficulties in draw, dilution, low utilisation and ground control over time. A systematic approach to regulating draw over the life of the panel is essential, especially if that approach is based on an optimisation methodology which is capable of estimating the impact of poor draw practice on key issues such as dilution and equipment utilisation.

2 TRENDS IN PRODUCTION SCHEDULE OPTIMISATION

Research into pit optimisation and scheduling has evolved into a reliance on LP-based heuristics, following on the early work in pit optimisation using graph theory for ultimate pit optimisation [Lerchs, 1965] and LP decomposition for long-term scheduling [Johnson, 1968]. Since the 60s, these methods have dominated the mining industry. While the original simplex-based algorithms have been greatly enhanced in terms of speed and flexibility, both the underlying assumption of deterministic data and a top-down hierarchical approach to optimisation have been retained. In contrast, scheduling applications in petroleum manufacturing, transportation and chemical industries have concentrated on scheduling using Mixed Integer Programming (MIP). Prior to recent computational and algorithmic advances, large-scale open cast mine scheduling problems were correctly perceived as being intractable when formulated as an MIP. In light of tremendous recent advances in MIP algorithms, computing power and parallel solvers, this is no longer the case. Recent advances in MIP-based scheduling applications are particularly relevant to underground scheduling applications such as draw control. The order relationships between sequencing of development and production and the order in which drawpoints are brought into production necessitates the use of binary (0/1) variables. This negates the option to apply traditional LP formulations, requiring the more computationally intensive use of MIP with Branch and Bound (B&B) searches.

Outside the mining industry, MIP-based production scheduling is the norm, especially in manufacturing [Pan, 1997], the chemical industry [Pinto, 1995] and petroleum [Lee. 1996]. Of particular importance are the methods these industries use to solve large-scale MIPs. MIP problems are difficult to solve [Papadimitriou, 1988], but a number of specialised algorithms based on Branch and Bound search, LP relaxation and cutting planes have been developed to efficiently solve these problems [Nemhauser, 1988]: the staircase structure of the scheduling MIP formulation must be exploited by a variety of means such as Special Ordered Sets (SOS), priority ordering, initial starting solutions, decomposition and preprocessing to aggregate constraints and fix variables. Extremely promising results have been realised using these techniques. Hane [I995], solved airline fleet assignment problems with over 22000 binaries via aggregation, benders decomposition and the Interior Point algorithm used for LP relaxation. Smith [2000] used block precedence relationships in open pit production scheduling to assign priority orders and fix integer variables, as well as a number of other B&B search strategies to solve ore blending problems with over 800 binaries within a cpu second on a 400mHz Pentium II.

MIP in production scheduling has only recently begun to be accepted in <u>mining</u>. Underground production scheduling applications include the scheduling of block cave draw points [Chanda, 1990] and slopes [Trout, 1995]. In surface mining, there have been only a few applications of MIP-based production scheduling [Caccetta , 1998; Graham-Taylor, 1992; Barbara, 1986]. Smith [1998] applied GP and MIP to the short-term production scheduling problem in large surface mines [Smith, 1999] and for blending and inventory control in phosphate mining.

Only in recent years have significant applications been used in underground mining. Trout [1995] used MIP to schedule ore production and stope backfilling at Mount Isa Mine. Chandra [1990] used MIP to schedule drawpoint production in a block caving operation. Muge and Pereira [1979], followed by Ribeiro [1982] applied dynamic programming to short-term production scheduling in sublevel stoping. Davis and Morrison *[1999]* discussed the use of Datamine's floating stope heuristic and its use in evaluating alternate stope configurations under conditions of geologic uncertainty, with the algorithm itself described by Alford [1995]. Another stope configuration heuristic, using the

Maximum Value Concept, was reported by Ataee-pour and Baafi [1999] Orvanic and Young [1999] used MIP and Special Ordered Sets of type 2 (SOS2) to optimise stope geometry by finding the consecutive sequence of blocks in a panel that yielded the maximum value.

All of these studies have concentrated on determining the optimum configuration of ore blocks for a stope. None address the issue of production scheduling. The only reported application to sublevel cave production schedule optimisation was a MIP model developed for the Kiruna mine by Almgren [1994]. In the Kiruna model, Almgren attempted to find an optimal schedule for the life of the entire mine, solving simultaneously for all periods. He encountered two difficulties: the resulting MIP was so large that an optimal solution could not be ensured, and uncertainty in the production system, geology and mixing rendered an optimal solution highly suspect. He concluded that single production period scheduling using a long-term objective function as suggested by Gershon [1982] was an acceptable alternative. Smith [2000] has demonstrated that a sequential MIP optimisation approach to substantial short-term production schedules is feasible even with modest computational resources.

Guest, et al. [2000] describe De Beers' MIP for block cave production scheduling currently in use in Koffiefontein. The De Beers scheduling system contains the primary components necessary for a block or panel scheduling system. The draw control system in use at Koffiefontein can be seen in Figure 4. De Beer's mineral resource auditing system (MINRAS) contains data on the panel contents, draw column and drawpoint status, and the availability of support facilities. This information is passed to the draw control scheduling program to determine the optimum draw schedule. The constraints currently used in the Koffiefontein MIP focus on block contents, block sequencing, and the availability of granbies. Once the draw schedule has been defined and implemented, the Koffiefontein vehicle monitoring system, Prodman dispatch, sends production details to the MINRAS system for use in updating the panel contents. Draw schedule optimisation then continues on a period by period basis.

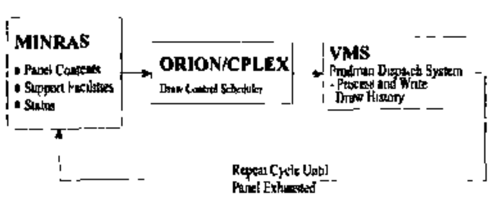


Figure 4 Data flow in the Koffiefontein draw control system

A more comprehensive De Beers scheduling system is being developed which includes a mixing module. The mixing module does not sit within the MILP, but is updated via the mine vehicle monitoring system. The production data is used to calculate the degree of mixing that has taken place as a result of the day's loading. This data is fed directly into the MINRAS database for use in determining the current panel contents for the next planning period. The MILP to determine the optimum draw schedule for each period with an objective based on maintaining a smooth, inclined dilution front. Guest, et al. [2000] report the usage of the mixing module and a time dynamic formulation based on NPV maximization.

3 GEOLOGIC UNCERTAINTY

Two sources of resource uncertainty limit the effectiveness of any scheduling system applied to caving. Mixing of the ore during draw has already been described and can only imperfectly be accounted for using empirical mixing models. This problem will be less severe in sublevel caving operations where there is limited height of draw and where control of blast fragmentation can be used to limit rilling. The other source of uncertainty arises from the spatial variability of the ore's characteristics - geologic uncertainty.

Geologic uncertainty's detrimental effect on mine planning and scheduling has been noted in recent articles which use simulated deposit models as input to the scheduling process. Ravenscroft [1992] describes the impact on production scheduling of deposit uncertainty using Conditional Simulation (CS). Dimitrakopoulos [1998] demonstrates that CS reproduces the underlying variability and spatial distribution of deposits and their use in open pit optimisation. Recent papers have used CS as a means of geologic risk assessment m: ultimate pit optimisation [Thwaites, 1998], long-term scheduling [Rossi, 1997], short-term scheduling and grade control [Blackwell, 1999; Dimitrakopoulos, 1999], NPV maximisation of stopes [Davis, 1999] and reduction of sulfur content in coal production [Costa, 1999]. While these studies have provided examples of the influence of geologic uncertainty as reproduced using CS, none have gone beyond basic sensitivity analysis. Smith and Dimitrakopoulos [1999] suggested a heuristic framework for quantifying uncertainty in short-term scheduling by coupling CS with Mixed Integer and Goal Programming (MIP, GP). The use of Stochastic Programming as a means of optimising production schedules in order to account for geologic uncertainty is discussed in Smith [2001].

Admittedly, at the current level of production scheduling as applied to block caving mere is little direct use for CS-based evaluations of geologic uncertainty: block caving operations are generally based on mining tons rather than grade or carats, an inevitable result of using an entirely non-selective mining method. Similarly, the technical level of mixing models do not account for the spatial distribution of planes of weakness and geologic boundaries, but for advances to be made in predicting caving, mixing and dilution, this technology will have to be supported with improved geologic models capable of accounting for uncertainty.

4 LIFE OF PANEL SCHEDULING

Production scheduling for open pit operations involves optimising the geometry and sequence of pushbacks and the production schedule for ore and waste over the entire life of the mine. This remains a largely trial-and-error methodology in which maximising NPV is balanced against a production schedule and an operationally feasible mine plan Time-dynamic scheduling over a rolling horizon can resolve the limitations of production scheduling, as all production periods are optimised simultaneously. In the rolling horizon approach, more emphasis is placed on finding the optimal solution for the most immediate series of production periods since these are deemed to be most critical and associated with greater certainty in terms of the state of the production system and stability of the market. More distant time periods represent longer phases of production and simplifying assumptions. In the caving application, these simplifying assumptions would include the availability of drawpoints and a static resource model. While this is the norm in manufacturing production schedules [Sethi, 1991] the few time-dynamic MIP applications in mining [Smith and Tao, 1993] have been limited to relatively small problems. Current practice aims at simultaneously optimising across all production periods for the mine's life with the aim of maximising project value. Both Guest [2000] and Almgren [1994] describe a time-dynamic formulation, but in practice both adopted *a* sequential optimisation approach. In reality, this approach increases problem size without adding to the quality of the solution; changes in technology, revised reserve estimates and shocks to the market inevitably negate any production plan extending beyond a relatively short time frame. Outside of the mining industry, production scheduling is optimised using a rolling front which consists of the minimum number of periods deemed necessary for making decisions relating to production and inventory planning; any changes in production capacity or demand are accounted for by «optimising starting from the current period. For block caving this is a methodology mandated by the dynamic nature of the resource model. In the context of surface mine production scheduling, long-range targets for production are still necessary in order to maintain a rational sequence of pushbacks that balance ore and waste removal [Tan, 1992]. In contrast, for the caving application, the production targets are based on minimising the deviation of the ore/waste horizon from the ideal surface. Thus, a minimum curvature surface becomes the long-range goal retained in each period.

5 DRAW CONTROL

Under ideal conditions of good fragmentation and draw without hangups, drawpoint scheduling would be a relatively straight forward process, especially if we could assume that the flow of ore is not spatially dependent on in-situ rock mass characteristics such as mineralisation and the ore/waste boundary. Unfortunately, draw rates and availability are not predictable over the life of a panel. Drawpoints in the active mining front are frequently closed or continued at reduced capacity due to unforeseen events such as early closure of drawpoints due to dilution and failures in the supporting materials handling system. Thus, scheduling is complicated by over producing in some drawpoints, bringing drawpoints on line at too early a date and by reassignment of production equipment. The long-term effect of deviations from an ideal schedule is that dilution levels become much more locally variable. As a result, a higher percentage of ore is left in the panel than would be the case if a smooth production front had been maintained. Therefore, the objective of draw control scheduling is to develop the means of minimising the impact of deviations from an ideal schedule in order to maximise ore recovery by minimising dilution and providing a stable mill feed.

For a given deposit realisation consisting of spatially variable mineralisation and structures, there is an optimal sequence of development and production that will allow the maximum extraction of ore from a panel. Heslop and Laubscher (1981) discussed some of the basic principals of block caving production, including: (1) lower the ore/waste interface as evenly as possible, (2) work all drawpoints simultaneously to achieve maximum interaction between drawpoints, (3) regulate drawpoint production as a function of the contained ore to avoid lateral migration of waste into isolated ore, and (4) maintain the ore/waste interface at a constant inclination so that the lines of drawpoints are depleted simultaneously as new lines come into production. The ore/waste boundary can be treated as a dilution front. In order to maximise the extraction of ore, this dilution front should be subparallel to the hangingwall and the plane of extraction of the system of retreating roadways, each of which terminates in a drawpoint. Excessive extraction from one or more drawpoints will result in increased curvature of the waste contact allowing a greater surface of dilution. As the curvature of the dilution front increases, so does the potential for early loss of ore. Poor control of the extraction sequence will become increasingly difficult to compensate for as the panel matures resulting in loss of reserves. Thus, scheduling of draw, retreat and development must always work towards the life-of-panel objective - maintaining a dilution front of minimum curvature.

6 FORMULATION

Draw control optimisation for a caving operation is a non-linear problem that cannot be solved simultaneously for all production periods. This characteristic arises from ore mixing changing the draw column contents throughout the life of the panel. The extent of this mixing Is m turn affected by mine production practice. The dependence of ore recovery on mining practice requires that the draw control problem be solved on a period-by-period basis. The long-term draw plan is implemented through the development a long-term objective which Is based on the Ideal draw strategy for the remaining life of the panel and the draw history of the panel. Figure 5 shows a snapshot of a block cave draw scheduling system as implemented after n production periods. Draw scheduling for the current period begins by transferring draw history data into a series of preprocessing modules. These modules determine the current contents of the individual draw columns based on an ore mixing model and the draw history of the columns. Additional information on the resources and support facilities available during the period (e.g. drawpoint availability, LHD availability, ore pass status, production targets) is also handled in preprocessing. This data is formatted as a parameter file for use in the optimisation module.

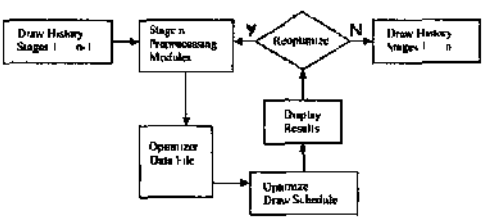


Figure 5. Procedural flow for sequential draw control scheduling.

The optimisation module then develops a draw plan based on the history of draw and the availability and maturity of drawpoints and the materials handling system. If limits on production prohibit the development of a feasible draw plan constraints on production must be relaxed and the problem resolved. Typically, this would involve increasing resource levels in the material's handling system, advancing the development schedule, or, at worst, relaxing production targets for mat period. The period-by-period planning process continues until a satisfactory draw schedule is produced. The resource model is then updated as mining proceeds and the entire cycle is repeated in subsequent planning periods.

The objective function seeks to minimise the deviation of column heights from the ideal surface (Figure 6). To achieve minimum curvature, production would come from the drawpoints lagging behind the ideal draw plan (designated by Xs) while all others would be closed during that scheduling period.

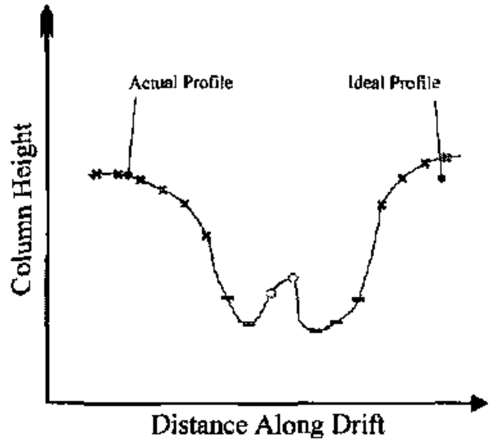


Figure 6 Ideal versus actual draw profiles

Now suppose that two of die allocated drawpoints have become unavailable due to ore hangup early in the production period. The LHD servicing the drift would not be able to achieve its draw. As a result, the LHD must draw from the suboptimal drawpoints to achieve its production target. The two best candidates are the two drawpoints marked by circles in Figure 6. These drawpoints break the continuity of the ore/waste contact by lagging behind their neighbours. Reallocating draw from the two hungup drawpoints to these two drawpoints will meet the LHD's production target while moving towards the ideal draw profile.

Order relationships have to be maintained to control die pattern m which drawpoint production is initiated. For example, at De Beers' Premier mine the pattern of initiating drawpoints is based on a fixed angle chevron pattern and a drawdown angle of 27 degrees (Figure 7). The orientation and angle of this pattern is controlled by precedence constraints between drawpoints in the same column (perpendicular to the direction of retreat).

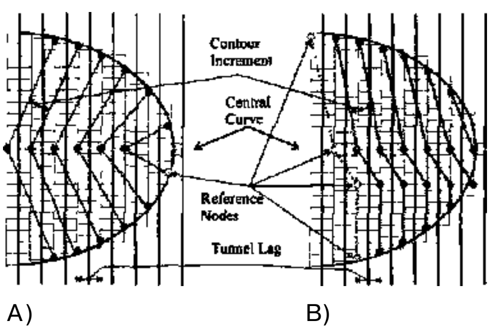


Figure 7. Ideal draw profiles as a) chevrons or b) variable configurations.

Any solution to tie drawpoint scheduling problem should ensure high utilisation of the materials handling system. As the result of non-ideal draw practice and the loss of drawpoints, solutions in the latter stages of a panel's life can identify that production should occur in fewer drawpoints or production tunnels than would be operationally acceptable, resulting in poor utilisation of tunnels and LHDs. In addition to minimising dilution, the solution should maximise equipment utilisation, even if this goal *is* secondary to resource recovery. Alternative methods are being explored which may accomplish this including: (1) a more elaborate constraint program as a front end to the M1LP, (2) including equipment utilisation constraints or (3) using hierarchical goal programming optimising on resource recovery first and equipment utilisation second.

7 CONCLUSIONS

There are clear advantages of using production scheduling as a major component of draw control in caving operations including: (1) minimising dilution and maximising resource recovery, (2) maintaining good cave propagation and fragmentation, (3) maintaining high equipment utilisation while avoiding bottlenecks in the materials handling system and (4) integration of the entire development and production system. De Beers and Kiruna are clearly moving İn this direction by adopting MIP-based scheduling. With the rising interest in applying caving systems to increasingly more challenging settings, the mining industry is investing in developing a generalised approach to draw control which İs built around scheduling.

There have been significant recent advances in production schedule optimisation both in the general industry and in mining. None of the scheduling methodologies currently available fully address complications associated with caving, in particular ore mixing and frequent loss of drawpoints. Both of these negate the use of time-dynamic solutions to the scheduling problem and require that production scheduling be carried out as an iterative sequence of single production period solutions. In each period, the history of draw is used to update the contents of a resource model while the current availability of the materials handling system and drawpoints constrains the solution to account for operational limitations and high system utilisation.

The clear objective of draw control is to minimise dilution over the life of the panel. Production scheduling aims at achieving low dilution by minimising the curvature of the ore/waste interface away from the İdeal surface. In each period, the history of draw is used to determine the difference between the height of columns in the resource model and the ideal level. The objective function then minimises the deviations in column heights as much as possible given the availability of drawpoints, production capacity maturity rules and limitations imposed by the state of the materials handling system.

The scheduling system envisaged herein requires a significant body of research İn a number of issues related to draw control and modelling. Accurately predicting the mobility of ore during caving and draw İs essential using an objective function based on modelling the position of the ore/waste interface. Otherwise, an "optimal" solution will be rendered useless due to the inaccuracy of the underlying resource model. Likewise, the collection of data on the production of each drawpoint has to be rigorously maintained and integrated into the production scheduling system. There are significant computational difficulties as well associated with formulating math programming models that include order constraints and scheduling of production equipment and development as large MIPs. This is a very difficult class of problem to solve requiring expensive solution engines and very powerful computers. In order to bring this technology to the mine site at the level of the production engineer will require research on efficient formulations and speed enhancing algorithms. While the application of MIP draw control is not a trivial problem, the technology needed to produce an effective first generation system is already on the shelf and the mining industry need only recognise the nature of the challenge and set about finding the solution.

ACKNOWLEDGEMENTS

The authors would like to acknowledge the support of the Director - Operations, Corporate Headquarters, De Beers Consolidated Mines and the Consulting Geotechnical Engineer Mining

In particular the contributions of Gert van Horn, Pat Bartlett, Kim Nesbitt, and Linda Hannweg of De Beers operations who assisted in the understanding and initial development of this work

REFERENCES

Alford, C Optimisation m Underground Mine Design *25th Application of Computers and Operations Research in the Mineral Industry Proceedings,* Brisbane, Australia, July 1995 AusIMM, 213-218

Almgren, Torgny 1994 *An Approach to Long Range Production and Development Planning with Application to the KirunaMme Sweden* PhD Thesis, Lulea University of Technology

Ataee-Pour and Baafi, 1999 *Slope Optimisation Using the Maximum Value Neighborhood (MVN) Concept 28th Symposium on the Applications of Computers and Mathematics in the Minerals Industry,* CSSM, Golden, Colorado, October 1999 493-502

Barbara , R W and R V Ramani , *Generalized muitipenod MIP model for production scheduling and processing facilities selection and location* Mining Engineer, 1986(February) p 107-114

Brady and Brown, *Rock Mechanics for Underground Mining,* 2[Rd] Ed Chapman and Hall, 1993

Blackwell, G , M Anderson, and K Ronson *Simulated grades and open pit planning - resolving opposed positions* In *27thAPCOM* 1999 CSM,Golden,CO SME

Caccetta, L, P Kelsey , and L Giannim *Open Pit Mine Production Scheduling* in *3rd Regional APCOM Symposium* 1998 Kalgoorlıe

Clianda, E C K *An application of integer programming and simulation to production planning for a stratiform ore body* Mining Sei and Tech, 1990 II p 165-172

Costa, J F , Koppe, J C , Dimitrakopoulos, R, and Ztnghano, A , *Incorporating uncertainty associated with prediction of geological attributes in coal mine planning* in *26th APCOM* 1998 London, 451-459

Davis, B M and J Mornsion *Enhanced underground design through optimization and risk assessment* In *27th APCOM* 1999 CSM, Golden, CO SME

Dimitrakopoulos, R *Conditional simulations and objective economic functions for grade control Implementations alternatives, pros & cons* SME Meeting, Denver, Co, March 1-3, 1999

Dimitrakopoulos , R *Conditional simulation algorithms for modelling orebod\ uncertainty in open pit optimisation* International Journal of Surface Mining, Reclamation and Environment 12,1998 p 173-179

Graham-Taylor, T *Production scheduling using linear and integer programming* in *AusIMM Annual Conf, State of the art - a product of 100 \ears of learning* 1992 Broken Hill, NSW AusIMM

Gershon, Mark *Heuristic Approaches for Mine Planning and Production Scheduling* Int J of Mining and Geo I Eng Vol 5(1),1-13 1997

Guest, A R , Van Hout, G J, Von Johannides, A, Scheepers, *LF An application of linear programming for block cave draw conti ol,* MassMın 2000, Brisbane

Hane, C A , et al *The fleet assignment problem solving a large-sca'e integer pi ogi am* Math Prog, 1995 70 p 211-232

Heslop, T Glen and Laubscher Dennis H 1981 Draw Control in Caving Operations on Southern African Chrysotile Asbestos Mines *Design and Operation of Caving and Sublevel Stoping Mines* Stewart, Daniel R (editor) SME, New York, 755-774

Johnson, T B *Optimum Open Pit Mme Production Scheduling, m Mining Engineering* 1968, University of California Berkeley p 120

Lane, K.F *The economic definition of ore cut-off grades in theory and practice* 1988, London Mining Journal Books I49pp

Lee, H , Pinto, J M , Grossman, I E, Park, S *Mixed-Integer Linear Programming model for refinery short-term scheduling of crude oil unloading with inventory management,* Ind Chem Res,35,1639-1641,19%

Lerchs, H and I F Grossmann, *Optimum Design of Open-Pit Mines* Transactions, C I M , 1965 LXVII1 p 17-24

Muge, Fernando and Pereira, Henrique Garcia. 1979 Short-term Planning in Sublevel Stoping Methods *16th Application of Computers and Operations Research in the Mineral Industry Proceedings,* Tucson, Arizona, October 1979 SME, 323-334

Nemhauser, G L and Wolsey, L A , *Integer and Combinatorial Optimization,* W dey Interscience, 1988

Ovanic, J and Young, D S 1999 Economic Optimization of Open Stope Geometry *28th Symposium on the Applications of Computers and Matkematics* in *the Minerals Industry,* Colorado School of Mines, Golden, Colorado, October 1999 Colorado School of Mines 855-862

Pan, C H *A study of integer programming formulations for scheduling problems* Int J Sys Sei, 1997 28(1) p 33-41

Papadimitnou, C H and K Steıglıtz *Combinatorial Optimization algorithms and complexity* seconded 1988 Dover 496 pages

Pinto, J M and IE Grossmann *A continuous time mixed integer linear programming model for short term scheduling of multistage batch plants* Ind Eng Chem Res, 1995 34 p 3037-3051

Ravenscroft , P J *Risk analysts for mine scheduling by conditional simulation* Trans Instn Min Metall (Sect A Min industry), 1992 lOl(May-August) p A104-A108

Rlbeiro, Luis Tavares 1982. *Dynamic Programming Applied to the Mining Sequence Optimization in a Sublevel Stoping Exploitation* 17th Application of Computers and Operations Research in the Mineral Industry Proceedings, CSM, SME, 494-499, 1982

Rossi, M E and B H Van Brunt *Optimizing Conditionally Simulated Orebodies with Whittle 4-D* In *Optimizing with Whittle Conference* Pp 119-128 1997 Perth

Sethi, S and G Sorger *A theory of rolling horizon decision making* Annals of Oper Res, 1991 29 p 387-416

Smith, M L and W Y Tao *Mine production scheduling for optimization of plant recover]' in surface phosphate operations* Int J Surface Mining, 1994 9 p 41-46

Smith, M L *Optimizing inventory stockpiles and mine pro duction an application of separable and goal programming to phosphate mining using AMPL/Cplex* Int J Surface Mining, 1998 p 61-64

Smith, M L and R Dimitrakopoulos *The influence of deposit uncertainty on mine production scheduling* Int J Surface Mining, 1999

Smith, M L *MIP solution strategies for short-term production schedules* Trans Inst M in Metall Section A mining ind, 2000 (submitted)

Smith, M L *Integrating conditional simulation and stochastu programming an application in production scheduling* APCOM 2001, Beijing, PRC, April, 2001

Smith , M L *Optimizing short-term production schedules in surface mining Integrating mine modeling software with AMPL/CPLEX.* International Journal of Surface Mining, Reclamation and Environment 12, 1998 p 149-155

Tan , S and R V Ramani *Optimization Models for Scheduling Ore and Waste Production in Open Pit Mines* In *23rd APCOM* 1992

Thwaites, AM *Assessment of geological un<.ertamt\ for a mining project ıtı27thAPCOM* 1998 London IMM

Trout, L P Underground Mine Production Scheduling Using Mixed Integer Programming In APCO

17th International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

High Technologies of the Underground Extraction of Kazakhstan Fields

Sh.A.Altayev, M.Zh.Bitimbayev & E.LZhumabayev
The Kunaev Institute of Mining, Almaty, Republic of Kazakhstan

ABSTRACT: In the underground extraction of ore deposits in Kazakhstan, full or partial backfill of the goaf is applied. In terms of market economics, goaf stowing is impractical. Thus, methods of synthetic roof fall of proof ores in mining have been designed.

1 INTRODUCTION

The method of synthetic slackening and roof falls of proof ores involves their deconsolidation by reactive solutions. For tins purpose, wells of 75 mm in diameter are bored in ragstones. Deconsolidating chemical solutions are forced under pressure into these wells.

Considerable deconsolidating and slackening of ore and barren sandstones have been obtained during experiments, and expenditure on explosives has been greatly reduced. This method will be tested and applied in one of the ore mines in Kazakhstan in 2001.

2 DOWN-HOLE MINING APPLICATION IN KAZAKHSTAN

It İs possible to develop a number of Kazakhstan friable ore fields, and also precious, non-ferrous and rare metals and uranium placer deposits applying know-how of hydraulic down-hole mining. Analytical investigations have been carried out and practical experience of hole know-how applied to the mining of different metal fields has been accumulated. Down-hole mining of friable leaden ores has resulted in a S-fold decrease in the production cost of 1 ton of ore in comparison to underground extraction.

Up to 45 % of Kazakhstan uranium field reserves are in composite geological conditions; therefore, these fields are developed applying down-hole know-how with the special methods of reservoir preparation for treatment and uranium leaching process intensification.

The engineering methods of fluid fracturing of soil parameter computation for the creation of synthetic aquicludes in uranium underground leaching, arguments the determination of intermediate shields, zones of permeability are designed. The processes of computation of physicochemical methods of underground leaching process intensification are introduced. Methods of calculus of optimal arguments of compound-structured reservoir exposure and development well construction are presented.

The special methods of uranium reservoir preparation and down-hole leaching intensification are designed on the basis of shield and hydrocurtain creation, colmatant injection, superwell construction, ultrasonic and electrical treatment of seams and screens, chemical decolmatation, effects of power surges, application of oxidants, magnetization, pulsating conditions, reversal of currents, etc.

The regularities of reagent current movement during filtering on bedding and crossing bedding of ore-bearing sandy and silty uranium reservoirs with different-granulated fractions and pore space, filled with clay formations grouted by montmorillomte and hydromicas are established. The links of seam protoxidation arguments with colmatauonal phenomena (gaseous, chemical) are recovered at different pH values of solution, overseaming head amount, and the availability of sulphate of calcium. The conditions of formation of aluminium and Ferri lactas hydroxides and the regularities of redepositions on the mobile geochemical barrier are determined.

The engineering methods (special and combined) of computation of opening up compound-structured uranium reservoirs to underground leaching are designed.

The idealized arguments of physics of the process of fluid fracturing of seams folded as friable sand-and-clay formations are given. The fluid fracturing computation method is designed and machinery necessary for its realization is determined. When this is applied in practice, the expected increase in uranium mined from 1 km" of reservoir floor space is 25-30 %, costs will be reduced 15-20 %, productivity will rise 20-30 %, and considerable savings of facilities, 60-80 million tenge, will be achieved annually. In addition, the ecological safety of e uranium mining will increase.

3 USE OF OLD MINES FOR VARIOUS PURPOSES IN KAZAKHSTAN

At several ore mines in Kazakhstan, there is practical expertise of using old mine workings for economic and manufacturing purposes. Analytical investigations of the use of old mines and ore mines for economically effective purposes are carried out.

Thus, capital investment is cut 1.2-1.5 times, and working costs are cut 1.5-1.8 times.

Methods of keeping long-lived old mine workings hot (for 50 years) and technologies of dumping parasitic wastes (uranium, mercury, arsenic) in old mine workings for infiltration into underground workings and mine waters are designed.

In the old mine workings of the "Youth" mine in the Karatau phosphorite basin and Belousovskii ore mine in eastern Kazakhstan, fungi, colours and vegetables are grown year round with large savings of facilities, particularly in winter time.

At mine № 39 in Zhezkazgan, old stops is used as stockyards and pigsties, and for beaver rearing and vegetable storage.

At one of the Zhezkazgan mines, there are plans to install machinery for copper dressing and discharge using a hydrometallurgical method in old underground mine workings. Thus, it is planned to increase the discharge of copper at the underground factory by 20 % m order to utilize solid washery waste for goaf stowing and to obtain a saving of about US$ 20 million annually.

Geotechnological methods of mining non-ferrous metals, both gold underground and heap leaching, are designed and tested practically in Kazakh ore mines. Experimental research is carried out and a project for mining zinc-lead ores by underground leaching is compounded in Tekeli ore mine. Thus, the withdrawal of zinc and lead compounds 80 % and considerable saving of facilities is obtained. In addition the necessity of keeping a large number of workers in underground conditions will be eliminated.

At Akchi-Spasskii opencast mine in Zhezkazgan, industrial trials of heap leaching of oxidated copper ores with large savings of facilities have been carried out. On the basis of the results of these operations, a heap leachmg lease of oxidated copper ores has been designed, with an annual output of 100 thousand tons of ore. Gold heap leaching from dressing tailings is developed at three ore mines in northern Kazakhstan - Aksu, Bestobe and Zholymbet. At Aksu ore mine, ladings from flotation occupy 90 ha floor space, where 7.3 million tons of tailings with 10.22 tons of gold and silver - 20.44 tons are stored; at Bestobe ore mme, these figures are 140.7 ha with 7.0 million tons of tailings containing 10.5 tons of gold and silver - 1.4 tons; and at Zholymbet ore mine - 88.5 ha, 9.0 million tons, 11.7 tons and 5.4 tons respectively.

For recovery of gold and silver from flotation tailings at Aksu, Bestobe and Zholymbet ore mines, dense alkaline cyanide leaching and sorbate concentration on highly alkaline anion exchanger, reactivation of gum and electrolysis of documentary reagents with precipitation of gold on the cathode are used. The technological system enables preliminary fine-grained tailings material pelletization (agglomeration) up to a fineness of aggregate of 15-30 mm with the use of 5 kg of cyanide and 2 kg of lime per ton of tailings. In order to maintain alkalinity at pH 10.5-11.5, CaO (0.6 kg per ton) is added when yarding tailings in stacks of heap leaching. As the resolvent, a mild solution of cyanide of sodium is applied. Cyanide solutions of such concentration dilute oxygen well and are fissile resolvents of gold. The desorption of gold is realized by eluting solution composed of thiourea and sulfuric acid in reverse-flow strings mixture. Electrolysis of the documentary reagent is done on carbonic-and-plumbago material by electrolytic cells witb. an output of 2 m per hour and 4% exit of gold on the current. Cathodic settling incineration is carried out by carbonic-and-plumbago fundamentals burning out at the temperature of 500-600°C. In the incinerated cathodic settling, the contents of gold compound 900-950 g per 1kg of the settling.

After leachmg the aqueous washing out of the heap leaching and tailings of leaching rendering stockpiles from cyanides up to the maximum allowable concentration by 25 % sulphite-bisulphite solution of ammonium is carried out.

In order to justify the building of a trial complex for the production of cathodic gold, a feasibility report on the development of gold heap leaching from tailings of flotation ores dressing at Aksu, Bestobe and Zholymbet ore mines was prepared by the D.A. Kunayev Institute of Mining. On the basis of the feasibility report, a detailed plan of production was prepared.

The resulting production is 1.6 tons of gold annually. The specific investments compound US$ 2.86 per gram of gold, and the cost price of the commodity output is 3.7 dollars per gram.

4 CONCLUSIONS

The modern technological possibilities and expenence gained allow 70-75 % recovery of gold from tailings dressing accumulated in the tailings storages of the Aksu, Bestobe and Zholymbet ore mines

17th International Mining Congress and Exhibition of Turkey- IMCET2001. © 2001, ISBN 975-395-417-4

The Assessment of Salt Production Drilling

A.Ersoy & T.Y.Yunsel
Department of Mining Engineering, Engineering Faculty. Çukurova University, Adana, Turkey

ABSTRACT: In this study, the salt production boreholes drilled by die Institute of Mineral Research and Exploration (MTA) for the Mersin Soda Industry in the Adana-Karakuyu-Arabali region were evaluated. The technical characteristics of die drilling equipment and cone (rock) bits used in the boreholes are explained. Caliche, claystone with siltstone, salt and anhydrite interbedding formations of the Pliocene-Messinian Age were drilled. Bentonite and natural water were used as the drilling mud. The density and viscosity properties of the drilling mud were measured continuously. The revolutions per minute (RPM) and weight on bit (WOB) were kept constant or widiin particular interval limits during drilling. This was due to the fact that the geological and mechanical characteristics of the drilled formations did not vary significantly. Although the salt reserve was not determined exactly, the underground salt thickness map suggests that there may be a huge reserve of salt. The salt was saturated with water and was pumped to the Mersin Soda Industry.

1 INTRODUCTION

The development and progress of a country depend on the correct assessment and scientific working of resources of raw materials both underground and at the surface. Drilling is an integral part of the extraction of underground resources.

Solution mining, as a descriptor of mining process, has been increasingly used in the literature over the past fifty years (e.g. Jessen, 1973; Boughten, 1973; Shock and Conley, 1974; Curfinan, 1974; Lefond, 1983; Folle, 1985; Saygili and Okutan, 1996; Haynes, 1997). A more accurate description of the general process could be "bore hole mining". The operation we are considering that of remaining a mineral value by drilling into ore body, circulating an extractive fluid and removing the mineral value. It is interesting note that the Frash process of mining sulfur is an outstanding example *of bore hole* mining This process dates back to the early 1900's. Certainly the many years of successful production of sulfur by the Frash process is a demonstration that bore hole mining has its place as a variable system.

The solution mining of salt offers an alternative to underground mechanical mining. Several operators are currently practicing a form of solution mining whereby a previously mined region is flooded, creating an underground sump, and tne brine is pumped to the surface for processing. The process involves drilling two or multiple wells into the salt bed and establishing a low pressure connection by directional drilling. Water is then injected into one well and the brine is collected from one or multiple production wells.

A little attention has been given salt production by solution mining method with the bore holes in Turkey. However, the Turkish trona deposit was discovered by the Mineral Research and Exploration Institute (MTA) in 1979 during coal exploration. The trona deposits are located approximately 15 km northwest of Beypazari in Central Anatolia, Turkey. Taking into account the whole features of the deposit, the Turkish trona deposit is the second largest natural sodium carbonate formation in the world. However, the production has not been made yet from the trona deposits.

Solution mining with the bore holes for salt production has only been applied for the study area in Turkey. The salt production from the area has been carried out by Mersin Soda Company since 1996. The study area is located in the Adana-Karakuyu-Arabali region. In the study area, caliche, claystone with siltstone, sandstone, and anhydrite interbedding formations are deposited. These formations are from the Pliocene-Messinian Age. The rocks contain salt of significant quality and quantity. The boreholes for salt production were drilled by the Institute of Mineral Research and Exploration for the Mersin Soda Industry.

In the study area, approximately ten boreholes and a total of 6000 m of drilling are completed each year. A total of 33 boreholes for salt production were evaluated, as illustrated in Figure 1. The salt formations were found at a depth of approximately 400-600 m. Fresh water is pumped through the pipe to the salt zone at the bottom of the hole. The salt zone is dissolved and the sail solution is taken out through the annular to the surface.

The purposes of the study include description of the drilling rig, examination of the bit operating parameters and the assessment and report of the salt production.

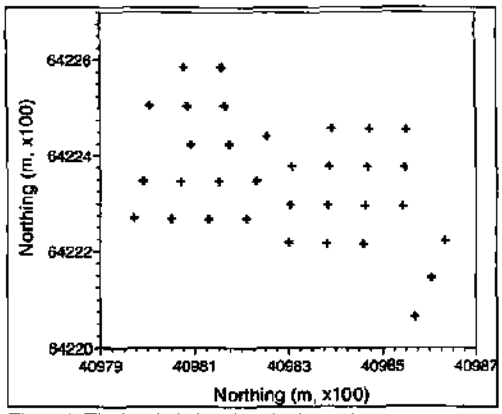


Figure 1. The borehole locations in the study area

2 THE DRILLING RIG

The drilling rig consists of a power pack, drilling assembly, mud pump and draw-works. Two diesel engines are used with the CF 2000 model drilling machine. These engines are reliable, readily available, durable and generally easy to service and maintain. The drilling rig is mounted on a lorry. These mobile units can produce very high outputs under suitable conditions, providing easy transport and using less time. Transmission of the developed power to various parts of the rig is achieved mechanically. In mechanical transmission, the power developed by each engine is gathered in a single arrangement, referred to by the term compound. The compound delivers the engine power to draw-works and a rotary table through roller chains and sprockets. In mechanical transmission, rig pumps are powered by the use of large bolts.

The drilling assembly contains a rotating table, kelly bar, drill pipe, drill collar, stabiliser and bit. The main function of the rotary table is to transfer rotary motion through to the kelly to the drill pipe, and eventually to the drill bit. The main function of the kelly is to transfer motion to the drill pipe, and transfer mud down to the drill pipes and to the bit. The swivel is installed above the kelly and its main function is to prevent the rotary motion of the kelly from being transferred to the drilling line. The drill pipe serves as a medium for the transmission of rotary motion to the bit and also acts as a passage for the mud. The drill pipes used in the drilling operations are 8.9 cm *{VA")* İn diameter and 14.1 kg/m (9.5 lb/ft) in weight. Drill collars are used primarily to put weight on the bit during drilling operations. The drill collars are 10.2 cm (4") in diameter and 74.4 kg/m (50 lb/ft) in weight. A total of six drill collars are used to provide weight on bit and to keep drill pipe in tension in the drilling line. The main functions of a stabiliser are to prevent buckling / bending of drill collars and to control drill-string direction. The drill bit constitutes the heart of the drill string and İs used to cut the rock for the purpose of making boles. Therefore, the proper selection and use of the bit have to be determined. The drill bit cuts the rock under the combined action of weight on the bit and rotary speed. Milling teeth of three-cone-type bits of different diameters are used in salt drilling. A simplified section of the drilling string is shown in Figure 2.

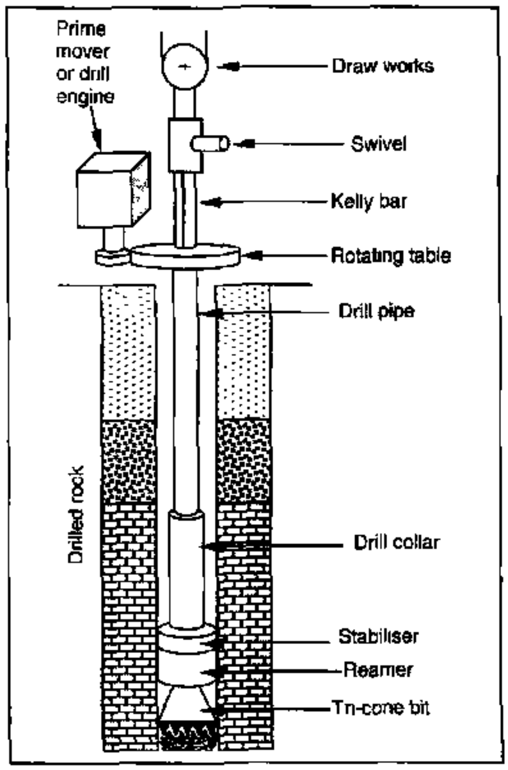


Figure 2. A simplified section of the drill suing.

A Gardner-Denver (FY-FXX) duplex mud pump is used to circulate huge quantities of drilling mud (161-372 gpm) down many hundreds of meters of the drill pipe, through small nozzles at the drill bit and, finally, up to the hole so that it reaches the surface again. Therefore, the pump must produce pressure in order to overcome the fnctional forces and move the drilling mud. The duplex pumps have double-acting pistons which create pressure on both the forward and backward strokes. The technical characteristics of the mud pump used in drilling are presented in Table 1.

The draw-works constitute the heart of the rig, enabling equipment to be run in and out of the hole. It also enables the driller, through the cat heads, to make or break the drill pipe, drill collars and other connections. General and detailed information about the drilling rig may be found in the literature (Rabia, 1985, Adams, 1985).

Tabie I The technical characteristics of ihe mud pump used in the rig

Piston diameter (inch)	Pressure (psi)	Stroke (spm)		Flow rate (jipm)		Max power (HP)
		Normal	Max	Normal	Max	
7 Vi	182			372	42Ü	
7	209			323	365	
614	242	62	70	277	313	53
6	284			235	266	
S'A	338			196	222	
5	409			161	182	

3 GEOLOGY OF THE AREA

The area is formed within the Adana Basın. The dominant hthology is caliche and clay formations with interbedded siltstone, sandstone, anhydrite and salt formations which are from the Upper Pliocene Messinian Age. The Upper Pliocene units of the Adana Basin contain geological records of a catastrophic event known as the "Messinian Salinity Crisis". These events, controlled by tectonics, affected all the sedimentary basins around the Mediterranean. The "Messinian Salinity Crisis" occurred as a result of the designation of the Mediterranean approximately 6 million years ago (Ogunc et al., 2000, Benson et al., 1991).

The following were cut during drilling in a characteristic well, caliche, claystone and clay with interbedded siltstone up to 300 m in depth; claystone, siltstone, sandstone and anhydrite alterations at a depth of 301-400 m; salt at 400-470 m; clay at 471-500 m; and salt at a depth of 501-590 m. A total thickness of approximately 130-150 m salt was drilled. All the drilling wells were stopped at a depth of 600 m from the surface. The characteristic geology of a drilling well is shown in Figure 3.

Geo-mechanical tests of the drilled formations have not been made due to absence of the core samples of the drilled formations. From time to time, core dnlling has been carried out. However, core recovery of the drilled formations is poor (core recovery < % 30) due to ^Konsolidation and softness of the drilled formations. Therefore, enough core samples were not obtained to perform many mechanical tests of the drilled rocks. Only average uniaxial compressive strength of claystone is determined to be 1.3 MPa.

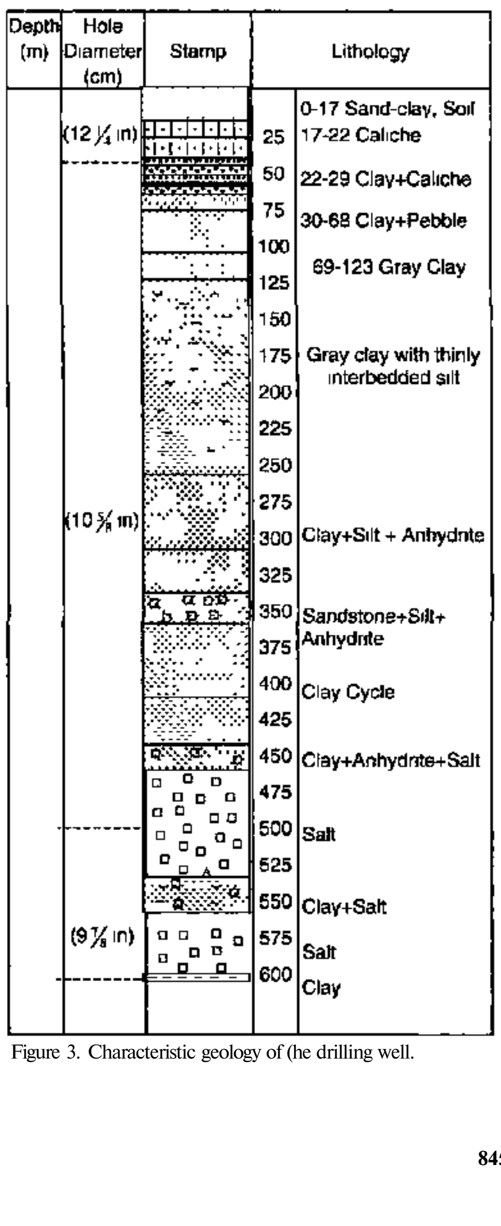


Figure 3. Characteristic geology of (he drilling well.

4 OPERATING PARAMETERS OF THE DRILLING BIT

A milled-tooth type of three-cone bit is used in the drilling, as illustrated in Figure 4. The bit consists of three equal-sized cones and three identical legs. Each cone is mounted on bearings which run on a pin that forms an integral part of the bit leg. Each leg is provided with an opening (for fluid circulation), the size of which can be reduced by fitting nozzles of different sizes.

Four different bit diameters were employed as shown in Figure 5. First, 50 m of the hole was drilled with a 31.2-cm (12.25")-diameter bit. The hole diameter was decreased when the depth was increased and, the 27-cm (10.625")-diameter bit was used to run for the longest part of penetration

The operating parameters of the bit can be identified as the rate of penetration (ROP), weight on bit (WOB), torque, and specific energy (SE). ROP is expressed in units of distance per unit time. It is one of the primary factors affecting the drilling cost. The main factors affecting ROP have been identified as the bit type or selection, WOB, RPM, torque, SE, bit hydraulics, formation or rock properties and fluid properties.

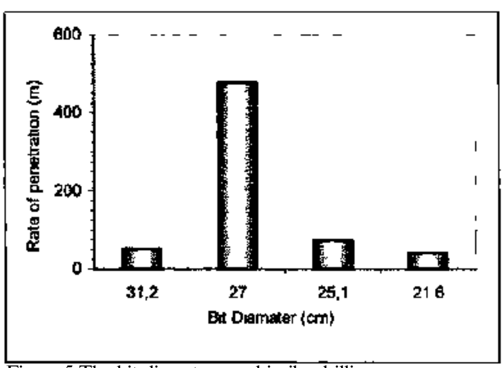


Figure 5 The bit diameters used in ihe drilling.

The daily rate of penetration is presented in Figure 6. ROP is high in the first part of the hole. When the depth of the hole increases, ROP decreases. As the hole depth increases, bit changes, drill string equipment to be run in and out of the hole and well problems take a long time, which causes a decrease In ROP. The cumulative ROP is shown in Figure 7.

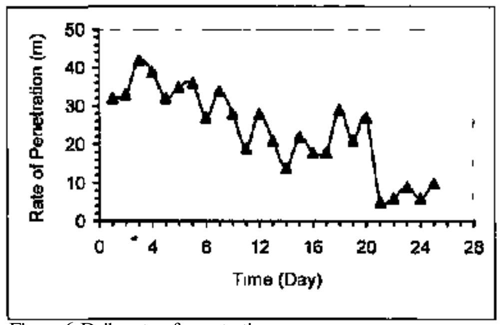


Figure 6 Daily rate of penetration.

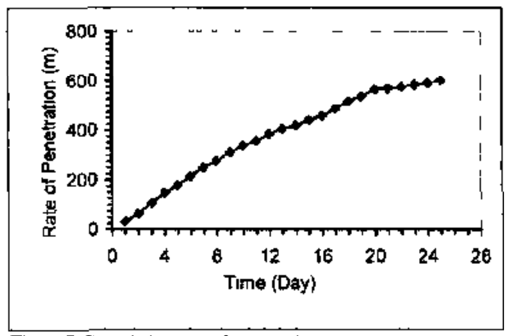


Figure 7 Cumulative rate of penetration

WOB, RPM, torque and SE were not measured continuously during drilling. This was due to the lack of measuring equipment in the rig and the shallow hole depth. WOB and RPM are independent parameters which can be directly controlled by the rig operator. These parameters were kept constant during drilling because the drilled formation does not undergo disturbances from tectonic activities, and the geological and mechanical properties of the rocks do not show significant differences. There is good homogenity and uniformity of the rock properties through the hole. The drilling assembly (the drill string) is approximately 7-8 tonnes in weight. 500-1500 kg of weight was applied on the bit. The other weight of the drill string was held by the hook of the draw-works.

The performance of the bit is significantly influenced by the RPM. An increase in RPM leads to reduction in the WOB required to attain a given ROP. However, the increased bit rotary speed results İn greater wear on the bit and may also cause chattering, microchipping and thermal cracking of the cutters of the bit. 120-180 rpm is applied to the bit according to the rock type drilled and the hole depth. This speed is continuously maintained and sufficient flush is provided in order to ensure the removal of rock cuttings from the hole.

SE can be defined as the energy required to remove a unit volume of rock (Teale, 1965). Thus, the drilling SE is a very significant measure of drilling efficiency, and it is directly compatible with cost per meter. The drilling SE can also be used as an indicator of bit condition.

Torque can be described as the turning force applied to the drill rod which leads to the bit rotating against the resistance due to cutting forces and friction. Torque control is performed for safety in drilling. Approximately the avergae torque value was 200 Nm during the drilling.

5 DRILLING MUD

Drilling mud is used to perform the following functions:

- Cool the drill bit and lubricate its teeth.
- Lubricate and cool the drill string.
- Control formation pressure.
- Cany cuttings out of the hole.
- Stabilise the well bore to prevent it from caving.
- Help İn the evaluation and interpretation of the well log.

In the study of salt production drilling, different mud types and properties were used at different lines of the well. In the first 100 meters, there is mainly natural soil and gravel, which have a high level of porosity. This can result in water-base mud leakage or fluid loss at the wall of the hole. At this depth, mud cake was formed on the wall using a mixture of water and bentonite. This cake helps to stabilise the wall of the hole, somewhat similar to the effect of adding a layer of plaster to interior house walls. The mud density, mud viscosity and annular mud velocity were frequently measured during drilling. The mud density was 1.1. gr/cm^3 and the level of viscosity of the Marsh Funnel was 40 sn for depths up to the first 100 meters.

From the depth of 100 meters to the salt zone in the well, water was used as mud (approximately 350 m in depth). At this line, claystone was the dominant formation. Thus, the clay cuttings of the bit were mixed naturally and circulated through the well. When the salt stratum was reached, the mud was saturated with salt. This process prevents the salt layers from being dissolved. At this stage, the mud weight was increased up to about 1.35 gr/cm^3 and the Marsh Funnel value was about 32-33 sn for viscosity.

The flow rate of the mud did not change often because tfiere was no significant difference in formation properties through the well. A flow rate of about 197 gpm was selected according to the technical specifications of the mud pumps. This flow rate was optimal for the removal of cuttings produced by the bit from the well to the surface. This requires a particular annular mud velocity. The mud velocity can be calculated by the following equations:

$$V = \frac{24.5 * Q}{D^2 - d^2} \qquad (1)$$

where,

V : Annular velocity (fpm).
Q : Flow rate (gpm).
D : Well diameter (inch).
d : Drill pipe diameter (inch).

$$V = \frac{24.5 * 197}{(10\tfrac{5}{8})^2 - (3\tfrac{1}{2})^2} = 14.63 \text{ m/dk (48 fpm).}$$

This is calculated for the bit diameter of 27 cm (105/8 inch), which was used in most of the drilling. This velocity is enough to carry the drilling detritus.

Deviation control was established for each 100 m in depth of the well. The deviation was measured by Eastman model magnetic Single Shot Instrument which records simultaneously, die hole inclination and magnetic north direction of an uncased hole at a single measured depth (or station). The instrument consists of an angle-indicating unit, a camera section, a timing device and battery pack In the drilling, significant deviation has not been recorded. Measured hole deviation was less than 2°.

6 SALT PRODUCTION

In the study area, two salt strata were determined. The first stratum was reached at a depth of 400-470 m in the well. The thickness of the strata is about 50-60 m. The second strata was found at a depth of approximately 500-590 m. The thickness of the stratum is 90 m. The total salt thickness is 120-150 m in the drilling wells.

Pumps are the hoisting mechanisms. The pump is located on the surface, force-feeding the injection fluid into the cavity at a pressure sufficient to raise the resultant brine to surface.

Top or annular and bottom injection was employed İn the study area (Figure 8). However, top injection was the most common type of brining operation used in the area. This approach involves installing a conductor or surface pipe through the unconsolidated surface overburden into the underlying bed rock by forcing a cement slurry down the inside of the casing. A follower plug, powered by mechanicat or hydraulic means, forces the cement to rise İn the annulus formed by the conductor pipe and the open hole. After the cement has been permitted to set, drilling of the hole is continued into the top of the uppermost salt bed and the mean string of casing set and cemented to surface in a manner similar to that used insetting the conductor pipe. The bit size is reduced so that it will be accommodated by the bore of main string Drilling of the hole is continued at this reduced diameter from the bottom of main string of casing to the bottom of the layered salt. The bore of hole then is equipped with a free hanging string of tubing extending from the surface to a point near the bottom of the well. Sometimes operators create a sump at the bottom of the well for the insoluble residues by initially operating the well in reverse (injecting water through the tubing).

The principal advantage of this top annular system is a simple uncomplicated well. Disadvantages are:

- The collection of insoluble around the bottom of the production well;
- The early exposure of large unsupported spans of roof rock which collapse, resulting in the beddmg and shearing of the production tubing;
- Dissolution of the upper portion of the bed by a blanket of insolubles;
- Low percentage of extraction;
- Generally low production rates (less than 3500-4000 liters per minute);

Bottom injection system involves the operation of a well similar in construction to a "top-injection" well, where the water İs continually injected into the tubing. In general, the bottom injection method produces a saturated brine at a lower rate of flow (750-2000 liter per minute)

The advantages claimed for this system are:

- A more uniform cavity shape;
- Less maintenance, due to greater "sump area" for insoluble and detrial material,
- Less blocking or plugging of the tubing with insoluble larger percentage of extraction.

Sometimes operators create a sump at the bottom of the well for the insoluble residues by initially operating the well in reverse (injecting water through the tubing).

A systematic (regular) method is applied to the determination of the drilling hole locations. The distance between the holes vanes from 30 m to 40 m (Figure 1). In this method of salt production, caves occur in the form of reserve cones underground. The range (effect area) of the two holes was united after a certain time. Then, inside pipes in casing are taken out of the holes, and fresh water is pumped to one hole, while the salt solution is taken from the other hole. Because, a cavity operating as single well has relatively low capacity for salt solution. The solution output increases considerably if two cavities are connected and the water is forced down one well and brine is removed from the other well. The salt production has been made in diameter of 30 m of the well. Approximately, pillar diameter was about 20 m between two holes. This distance is left to prevent formation subsidance. Thickness of salt bed is of prime importance because it determines the yield of a well and consequently the economics of the process. Furthermore, as rock salt provides a stronger roof structure, larger cavities can be developed which prolongs the life the rock salt wells.

Host minerals of the salt, such as clay, anhydrite, quartz and feldspar sink to the base of the caves. The inside pipes are shortened from time to time to prevent stoppage of the host minerals. Water comes to the upper end of the cone, whereas the salt solution goes down. The concentration increases continuously on the side wall of the cone.

Contour mapping and three-dimensional graphs, created using the Surfer 7.00 computer package, are very useful for the spatial continuity analysis of large quantities of data (Golden Software, 1999). A three-dimensional graph and contour map of the thickness of the first salt stratum underground are presented in Figures 9 and 10, respectively. Similarly, Figures 11 and 12 show the second salt stratum. Consequently, these figures exhibit that the salt stratum underground is more or less regularly distributed. This allows easy production and low production costs.

In the study area, the quantity of the salt reserve was not determined. However, the distribution of the salt thickness suggests that there may be huge salt potential (Figures 9-11). 500 million tons salt reserve has been estimated İn the area. It should be mentioned that the reserve for the salt formations may be esbmated using contour map of the salt thickness (Figures 10 and 12). This is out of the scope of this study. Produced salt solution has been pumped to Mersin Soda Industry by the pipe line of 46 km in length.

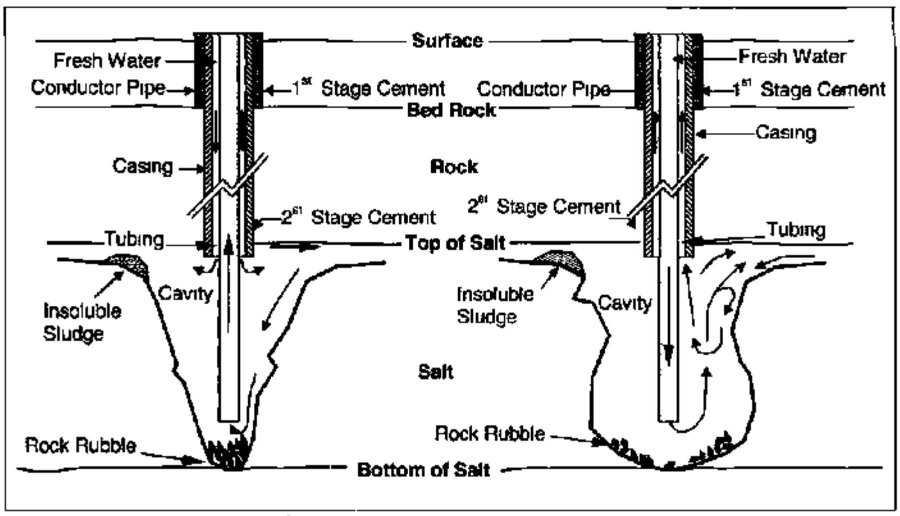


Figure 8 Top and bottom injection for salt production methods with bore hole systems

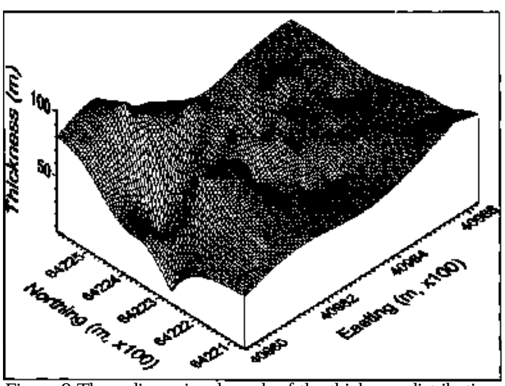


Figure 9 Three-dimensional graph of the thickness distribution for the first salt strata

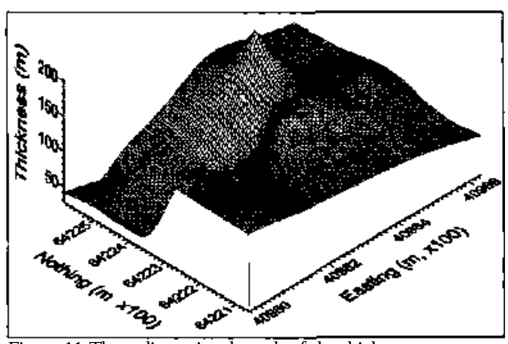


Figure 11 Three-dimensional graph of the thickness distribution for the second salt strata

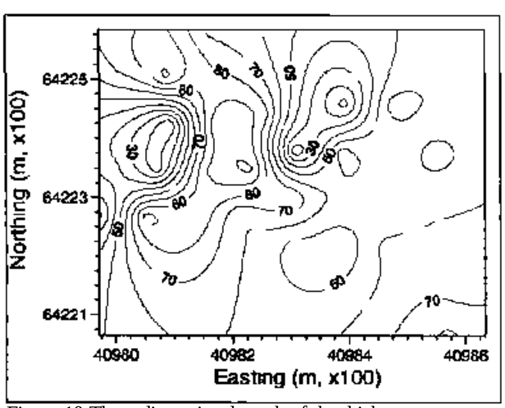


Figure 10 Three-dimensional graph of the thickness distribution for the first salt strata

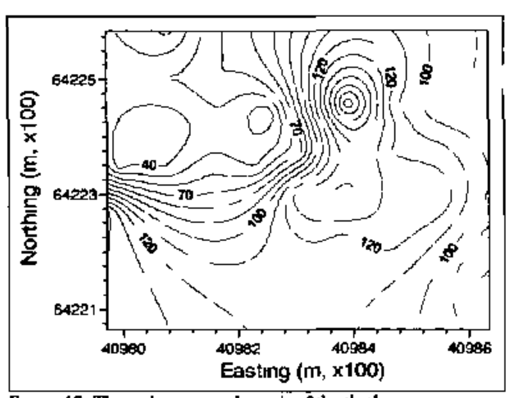


Figure 12 Three-dimensional graph of the thickness distribution for the second salt strata

7 CONCLUSIONS

The main conclusions of the study may be summarised as follows.

1. A total of 33 boreholes for salt production with solution mining method were examined.

2. Caliche, claystone with siltstone, sandstone and anhydrite interbedding formations were drilled through the well.

3. The salt zone was reached at a depth of 450-600 m in the well.

4. A characteristic milled-tooth type of the cone bit was used due to soft and unconsolidated formations were drilled.

5. The bh operation parameters such as WOB and RPM were kept constant or within particular interval during drilling. Approximately the average value of WOB was 1000 kg and the average value of rotational speed was 150 rpm.

6. Drilled formations did not show significant differences İn terms of geological and mechanical characteristics of the drilled rocks. Therefore, the operating parameters of the bit were not changed frequently during the drilling.

7. ROP was decreased, when the hole depth increased.

8. Bentonite and water were used as drilling mud up to the salt zone. However, saturated water with salt was applied as mud in the salt zone. The density of the saturated water with salt was about 1,38 gr/cm^3.

9. The measured hole deviation was less than 2°.

10. The salt strata was dissolved by pumping of water and produced through annulus to the surface.

11. Contour maps and three-dimensional graphs of the salt thickness suggest that the salt underground is distributed more or less regularly.

12. One of the main problems during drilling is that clay cuttings stick to the bit and this obstructs the rotation of its cones due to swelling.

REFERENCES

Adams, N.J. 1985. Drilling engineering, a complete well planning approach. *Penn Well Press,* USA.

Benson, R.H., El Bied, K.R. & Boraduce, G. 1991. An important currant reversal (influx) in the Rifion Corridor (Morocco) at the Tortonian Messinian Boundary: The end of Tetths Ocean. *Palaeoceanography.* 6:164-192.

Boughten, L.D. 1973. Solution mining well-completion services and techniques. *SME Mining Engineering Handbook. 2,* Editor: Cumins, A.B., The American Institute of Mining, Metallurgical and Petroleum Eng, Inc., New York.

Curfman, R.L. 1974. Solution mining project. *In Solution Mining Symposium.* Editor: Apian, F.F. et al. :173-184, New York.

Folle, S. 1985. Geological conditions for the construction of salt caverns. *Salts & Brines' 85, Proceedings of The Symposium-Solution Mining of Sails and Brines.* New York.

Golden Software Inc. 1999. Surface Mapping System, version 7.00. USA.

Haynes, H.W.Jr. 1997. Solution mining of trona. *Marcel Dekker Inc.* 21: 357-394, New York:.

Haynes, H.WJr. 1997. Solution mining of Wyoming trona. *Proceedings of the 1997 SPE Rocky Mountain Regional Meeting,* May 18-21 1997, USA.

Jessen, F.W. 1973. Specialized underground systems: Solution Mining. *SME Mining Engineering Handbook.* 2, Editor: Cumins, A.B., The American Institute of Mining, Metallurgical and Petroleum Eng. Inc., New York.

Lefond, S.J. & Jackoby, C.H. 1983. Salt in industrial minerals and rocks. Editor. Lefond S.J. 2: 1119-1150.

Ogrunc, G., Gurbuz, K. &. Nazik, A. 200Ö. Evidences about Messinian salinity crisis in the upper Miocene-Pliocene sequence of the Adana Basin. *Yerbilimleri: Bulletin of Earth Sei. Application and Research Centre of Hacettepe University:* 22: 183-192.

Rabİa, H. 1985. *Oilwell drilling. Engineering principles and practices.* Oxford: Graham and Trotman Limited.

Saygili, N.-S. & Okutan H. 1996. Application of the solution mining process to the Turkish trona deposit. *Hydromelalturgy.* 42: 103-113.

Shock, D.A. & Conley, R. 1974. Solution mining- its promise and its problems. *In Solution Mining Symposium.* Editor: Apian, F.F. et al. : 79-97, New York.

Teate, R. 1965. The concept of specific energy in rock drilling. *Int. J. ofRockMech. and Min. Sei.* 2:57-73.

Tokgonul, F. & leacan, 1. 2000. Yenice tuz sondajları 199 yılı sondaj faaliyet raporu (unpublished). *Maden Tetkik ve Arama Bölge Müdürlüğü.* Adana.

17th International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

Ore Deposit Mining with Goaf Stowing in Kazakhstan

V.Muzgina
"Gornoe Büro " Ltd., Almaty, Republic of Kazakhstan

ABSTRACT: This paper presents the historical aspects and modem conditions of the filling method used to exploit deposits of non-ferrous metals at mines in Kazakhstan.

1 INTRODUCTION

Kazakhstan is a republic with a highly developed mining industry. Many deposits of non-ferrous metals (lead, zinc, copper) are part of our national resources. Unfortunately, many such deposits involve difficult mining and geological conditions. Mining science and practice has confirmed the expediency of the filling method used for ore extraction from such deposits. That is why in the last 20 years of the 20^{th} century, the filling method has become common in mining lead, zinc, copper and other non-ferrous metals in Kazakhstan.

2 HISTORICAL REVIEW

Filling operations were used first in Kazakhstan on a large scale at the beginning of the 1960s, when the first filling complex was built at Tekely mining and processing integrated works (Almaty region). The Tekely lead and zinc deposit is a thick steep (70-80°) ledge, located in the mountain massif Dzhungar Alatau. Its characteristic properties are high rock pressure and the tendency of enclosing rocks to self-ignition because of the active form of the pyrites present. Firstly, the deposit was mined by caving system, but after a large spontaneous fire in 1958-1959, a solution was proposed to change the mining system and further exploitation of this deposit continued with the filling method. Planned production at the Tekely filling complex was 230 thousand m^3 a year. The filling mixture contained crushed stone - 1400-1500 kg/m\ cement - 200-250 kg/m^3 and water - 450-500 l/m^3. During this filling complex exploitation, disadvantages and practical problems were brought to light, and the following filling complexes at other deposits were built taking these into account. At the end of the 1960s and the beginning of the 1970s, filling complexes with productivity of 100-500 thousand m^3 (and more) a year were built in Kentau (southern Kazakhstan region), Leninogorsk and Zyryanovsk (eastern Kazakhstan region). Since 1972, filling complexes have begun to operate in Zhezkazgan - the center of Kazakhstan's copper production, where the most copper deposits in Kazakhstan and one of the biggest copper deposits in CIS countries are located.

In 1968, at the Institute of Mining of the Kazakh Academy of Science, a new scientific research laboratory was set up to cany out investigations of filling problems. Such laboratories were also set up at otfier scientific research institutions İn the Republic of Kazakhstan. It was a period of intensive introduction of different variants of the filling method, different new compositions of filling mixtures and study of properties of man-made massifs in underground conditions.

3 MODERN CONDITION OF FILLING METHOD USED IN KAZAKHSTAN

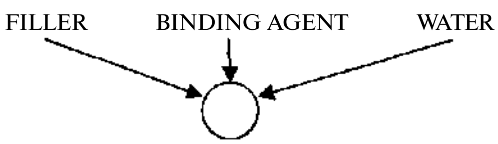
At the beginning of the 1990s, modern political and economic tendencies were the cause of great changes in the mining industry of the Republic of Kazakhstan. In market economic conditions many mining enterprises had problems in financing their activities. In as much as profitability is now the main index of expediency in mining operations (excluding social and political factors, which were of great importance in the socialist economy), some mining enterprises stopped production because of very high product cost and the impossibility of selling their products at a profit in order to cover expenses. Some mining enterprises decided to change the mining system to exclude filling operations because of the great expense. But a number of mining enterprises, having very tough mining and geological conditions, could not stop filling operations completely. That is why they tried to decrease the cost of filling operations and make cheaper filling mixtures. Investigations of filling problems are now carried out practically in two ways:

- improvement of technology of filling operations;
- working out filling mixtures with industrial solid waste.

The first method includes the creation of new types of filling equipment of high productivity and reliability and requires additional capital costs. However, worked-out new technological systems of filling massif forming, which have been suggested by our scientific researchers (Yedilbayev et al., 1999), ensure cement İs saved without decreasing die necessary technological characteristics of a man-made massif, while at the same time increasing the economic effectiveness of filling operations.

The second method consists of working out new filling mixture compositions with different solid waste utilization. This method also has an ecological aspect - decreasing the negative effect of the mining industry on the environment.

As is well known, filling mixture contains the components shown in Figure 1.



Figure! Composition of filling mixture.

As a filler, specially mined crushed stone and sand or different solid waste (shifted broken rock, ore tailings and so on) are used.

As a binding agent, as a rule, structural cement is used. To decrease cement consumption, active mineral admixtures are used: ash and metallurgical slag, which have some binding activity.

The majority of filling complexes of non-ferrous metallurgy in Kazakhstan use ore tailings as their raw material supply because of the following:

- large volume of ore tailings, both current and stored, in tailings piles, ensuring stable productivity of filling complexes;
- relatively stable granularity of ore tailings, having a significant effect on compressive strength of man-made massif;
- possibility of transportation of ore tailings from day surface up to worked-out space through pipelines.

Our investigations showed that the mineralogical composition of ore tailings has a significant effect on the compressive strength of a man-made massif. That is why cement consumption in different filling complexes is not the same, and to substantiate necessary cement consumption we may only work on the basis of experimental tests.

It is possible to use shifted broken rock to fill worked-out space. But in underground mining of deposits by chamber-and-pillar systems, the volume of such rocks is only 30-40% of worked-out space volume; therefore, they may be used only as admixture to save cement in chambers in the second stage of extraction

Introduction of new technologies of processing of ores causes the emergence of new types of solid waste. Consequently, heap leaching of copper ores İn Zhezkazgan required investigations with regard to the utilization of solid waste in mis process. The investigations showed that filling mixtures with waste from copper ore heap leaching have high indexes of compressive strength and may be successfully used m filling operations as filler.

As is well known for the preparation of filling mixtures, structural cement is used. It is now very expensive and the use of such a binding agent decreases the economic efficiency of filling operations. Investigations carried out (Yedilbayev et ai, 199?) showed that it is possible to make high quality cement for filling operations and also for civil-engineering purposes from local materials and solid waste. It İs one of the ways for the mining industry of the Republic of Kazakhstan to decrease the costs of filling mixtures at filling complexes.

To decrease cement consumption, some active admixtures may be used. Investigations carried out showed that boiler-room ash admixture in the range of 7-15% allows an increase in the compressive strength of the filling composition by 40% with a decrease at the same time in we cement content by 10-12%.

So we have a wide range of industrial solid waste, and it is possible to make filling mixtures with complexes using solid waste as shown in Figure 2.

Complexes of solid waste allow a wide supply of raw materials for filling operations, decreases the negative effects of solid waste - accumulation in dumps, tailings piles, ash dumps etc. - on the environment, and saves very expensive material, cement, when making filling mixtures.

CONCLUSIONS

The mining industry in Kazakhstan now has problems in financing the activity of mining enterprises. Underground ore mining and goaf stowing have become very expensive in spite of the low level of mining losses and ore dilution which are characteristic of these mining systems. To increase the economic indexes of these systems, İt İs necessary to improve the technology of filling operations and to use complexes of solid waste to make filling mixtures. In addition, of course, mining systems with goaf stowing must be used when mining useful minerals that are in great demand in the world market and have a high selling price.

FILLER

Ore tailings (the main component)

Admixtures

Shifted broken rock | Solid waste of heap leaching | Other types of solid waste

BINDING AGENT

Structural cement | Special filling cement (local materials and solid waste)

Admixtures

Ash | Milled metallurgical slag | Other active admixtures

Figure 2. Scheme of complex solid waste used when making filling mixtures.

REFERENCES

Yedilbayev, AI., Mıızgma, V.S. & Omirzakova, EZh 1999. Effectiveness of filling massif forming with support-stiff layers. Proceedings of *MPES-1999,* Dnepropetrovsk. Ukraine' 339-341

Yedilbayev A I., Aldambergenov I) A & Muzgina, V.S 1997 Using local materials and solid waste for special mine-fill cement production. Proceedings of the *15^th^ Mining Congress of Turkey* 141-142.

17th International Mining Congress and Exhibition of Turkey- IMCET2001 ,©2001, ISBN 975-395-417-4

The Effect of Design Peculiarities of the Elastic Liner of a Hoisting Machine on the Durability of Rope and Liner

V.P.Franchuk
National Mining University of Ukraine, Dmpropetrovsk, Ukraine
V.V.Franchuk
Technological University ofPodillia, Khmelnitskiy, Ukraine

ABSTRACT: This paper presents a comprehensive approach to the study of rope and liner. The basic principles of liner material selection and the main geometric dimensions of the liner were worked out on the basis of investigations conducted on the stress-strain state of the liner. Using the modern concepts of interaction of bodies with a movable contact point, the mechanism of stress accumulation and the reasons for the increased wear of the liner were determined. On the basis of these investigations, the principles for calculating the dimensions of the liner material along the drum generatrix were established. As a result of these studies, recommendations for material selection as well as for the calculation of geometric characteristics of the drum liner of the hoisting machine are made.

1 INTRODUCTION

The interaction of the rope of a hoisting machine with the drum surface İs accompanied by wear of both the rope and the liner. The wear of the rope and the liner is dependent upon many factors, such as the value and the character of the forces of interaction of the contacting surfaces, as well as their relative displacements (Franchufc & Franchuk, 2000).

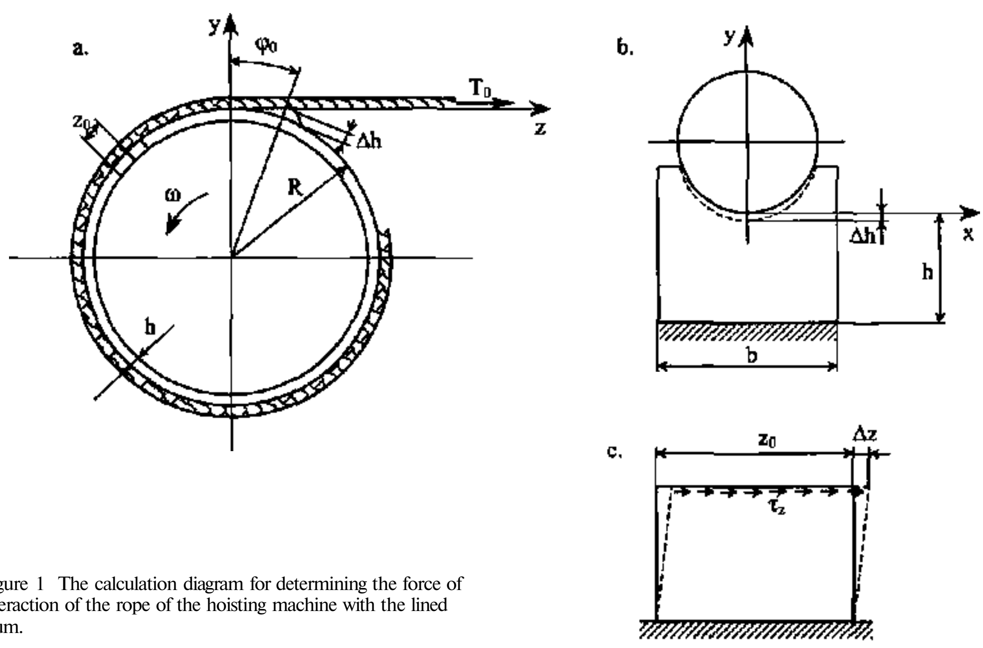


Figure 1 The calculation diagram for determining the force of interaction of the rope of the hoisting machine with the lined drum.

2 VALIDATION PROCEDURE

The most intensive slip of the rope on the liner surface occurs in the zone adjacent to die point of the running-on of the rope over the drum This displacement takes place due to longitudinal elongation and torsion of the rope the angle of deviation. The value of these displacements depends on the rope characteristics, the design and the elastic parameters of the drum liner. If the liner is elastic enough, the slip of die rope on the liner surface decreases. At the same time, if the liner material has a sufficiently large Poisson coefficient, the high give of the liner results in the formation of some convexity on the liner surface, which can be called the "deformation wave" (or "deformation roll"). At the beginning of the motion of the hoisting machine drum, growth in the deformation wave (in front of the rope running-on point) takes place. When this wave reaches a certain value, further growth stops. The accumulation of material in die deformation wave is balanced out with die slip of the surface of the rope and the liner. Thus, there is intensive wear of the rope surface and the liner surface.

The aim of this paper is to investigate the nature of the rope-liner interaction. The present investigation should result in the design selection and calculation of die elastic liner of me hoisting drum.

2.1 *Details of the example problem*

Let us consider the interaction of the steel rope with the drum of the hoisting machine. The drum is lined with elastomer-type elastic material. Figure 1 represents the calculation model for determining the force of interaction of the rope and the drum liner. The drum of the hoisting machine with radius R rotates at an angular velocity ω. The hoisting rope Is loaded with force T_0. In order to determine normal loads q acting from the side of the rope on the liner, the pulling force T_0 of the rope becomes a major factor. The value of the tangential load is also influenced by the forces of engagement of the liner material and the rope. The dependence between shear stresses T and normal stresses p on the surface of the interaction of the rope and the liner will assume the form (Novikov et al. 1978):

$$T \wedge p \quad W+\beta_0 x.-a_0 p \qquad (i)$$

where % = —— is the ratio of the relative velocity of

motion of the contacting bodies to the absolute velocity of the rope displacement; 8o, ßo, Xo are the experimental coefficients.

Figure 2 presents the plot of dependence of the shear load T, on the relative velocity of motion of me

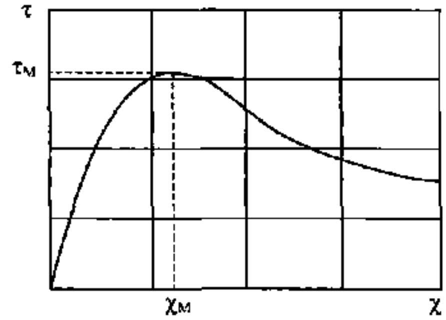


Figure 2 The dependence of the shear load T on the relalive velocity of motion of the drum and the liner *X*-

drum and the liner y- The plot has a rising part and a falling part, and the maximal value of engagement force of the rope and die liner is obtained at the relative velocity /„. The value of this force, as well as the corresponding relative velocity V, can be calculated from the expression:

$$\chi_M = \sqrt{\frac{\lambda_0 p}{2\delta_0 V}} \qquad (2)$$

or

$$v_{12M} = \sqrt{\frac{\lambda_0 p V}{2\delta_0}} \qquad (3)$$

which is obtained by investigation of expression (1) to the extremum.

When the liner is continuous, die running-on of the rape over the drum is accompanied by the presence of the deformation wave (or roll) and the constant slip of the surfaces of the liner and the drum. The velocity of relative motion x(ˇi2) can correspond to the rising or the falling parts of the engagement curve (Figures 2, 3). The relative motion on the rising part of the curve is accompanied by creep and leads to insignificant wear of the surfaces. When the relative velocity is more than y_Mp the slip occurs, which is accompamed by considerable wear of the liner and the rope.

2.2 *Analysis procedure*

The relative velocity of the slip of the rope and the liner In the running-on zone can be determined on the basis of the calculation model shown in Figure 1. Let us assume that element deformation takes place along the plane surface in the direction of the y-axis. The lateral surfaces (in the direction of the x-axis) are free. During rope winding, there is an accumulation of deformation in the direction of the z-axis. At the same time, considering the material anisotropic, we obtain:

$$\varepsilon_y = \frac{\sigma_y}{E}, \quad \varepsilon_x = \nu\varepsilon_y, \qquad (4)$$
$$\varepsilon_z = \nu\varepsilon_y, \quad G = \frac{E}{2(1+\nu)}$$

Here, E and G are modules of elasticity in compression and in shear; £K, £j, e£ are relative deformations in the direction of the corresponding axes; v is the Poisson coefficient

Under constant rope pulling, the liner material having the deformation wave height Ah at the initial moment will move during the rotation of the drum to the angle (po (Figure la), m relation to the rope to the value

$$\Delta z = R\varphi_0\varepsilon_z$$

or, taking into consideration (4):

$$\Delta z = R\varphi_0\nu\varepsilon_y = R\varphi_0\nu\frac{\Delta h}{h} \qquad (5)$$

Then the relative velocity of the slip is:

$$v_{12} = \frac{\Delta z}{\Delta t} \qquad (6)$$

and the time of rotation to the angle (puis:

$$\Delta t = \frac{\varphi_0}{\omega} \qquad (7)$$

where to is the angular velocity of the drum rotation.

From equations (5), (6), (7) we will have:

$$v_{12} = \frac{R\nu\omega\Delta h}{h} \qquad (8)$$

Since

$$\frac{\Delta h}{h} = \varepsilon_z = \frac{\sigma_z}{E} \approx \frac{q}{d_k E} = \frac{T_0}{ERd_k}$$

then from expression (8) we obtain:

$$v_{12} = \frac{T_0\omega\nu}{Ed_k} \qquad (9)$$

where d^ is the rope diameter.

If we compare the value vu obtained from expression (9) with the value obtained from expression (2), we can determine at what section (rising or falling part) of the engagement curve the *system* works (and, correspondingly, determine the character of engagement and wear). For a liner composed of separate elements placed along the drum surface, die accumulation of deformation occurs only in the direction of the z axis. Then maximal shear stresses (Fig. 1 ,c) will equal:

$$\tau = G\gamma = G\frac{\Delta z}{h} \qquad (10)$$

Taking into account that:

$$\varepsilon_y = \frac{\Delta z}{z_0}$$

and, in turn:

$$\varepsilon_z = \nu\varepsilon_y = \nu\frac{\sigma_y}{E}$$

we obtain:

$$z_0 = \frac{E\tau h}{G\nu\sigma_y}$$

If we take into consideration equation (4), we will have:

$$z_0 = \frac{2(1+\nu)\tau h}{\nu\sigma_y} \qquad (11)$$

In order to decrease friction and wear of the rope and the liner, it is necessary that shear stresses should be less than critical. This will ensure the engagement of the surfaces and ensure that the engagement will not break down.

Assuming:

$$p = p_{mid} = \sigma_y = \frac{T_0}{Rd_k} \qquad (12)$$

and,

$$v = R\omega$$

after transformations, we will have the dependence for determining the ultimate length of the liner material, ensuring relative motion of the surfaces of the liner and rope wimin the creep limits:

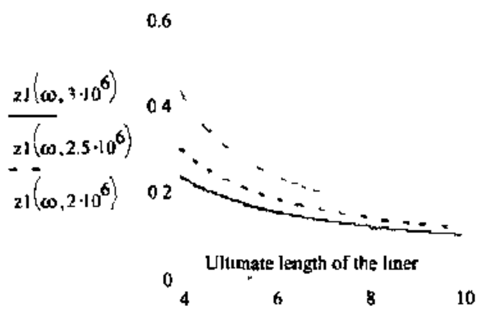


Figure 3. The dependence of the ultimate length of the liner *material on* the angular velocity *of the drum rotation*

$$z_{0f} \le \frac{2(1+\nu)}{\nu} \cdot \frac{\sqrt{\dfrac{\lambda_0 T_0}{2\delta_0 d_k \omega}}}{\dfrac{3}{2}\lambda_0 \dfrac{T_0}{d_k} + \beta_0 \sqrt{\dfrac{\lambda_0 T_0}{2\delta_0 d_k \omega}}} \cdot h \qquad (13)$$

3 RESULTS

Let us analyse the results obtained- Figure 3 demonstrates the plot of dependence of the ultimate length of the liner material on the angular velocity of the drum rotation. If the angular velocity increases, the ultimate length of the liner material somewhat decreases The same phenomenon takes place during the increase of the ratio $\frac{T}{d_k}$. It is clear from equation (13) that the value z,f is proportional to the line thickness and does not depend on the drum diameter.

A combined liner was created and installed (Figures 4, 5) for a one-drum hoisting machine. This liner consists of alternating strips of polyamide elements and rubber strips. The polyamide elements have a groove under the rope. The rubber strips were made of worn-out truck tyres. These strips have no groove under the rope is intermittent.

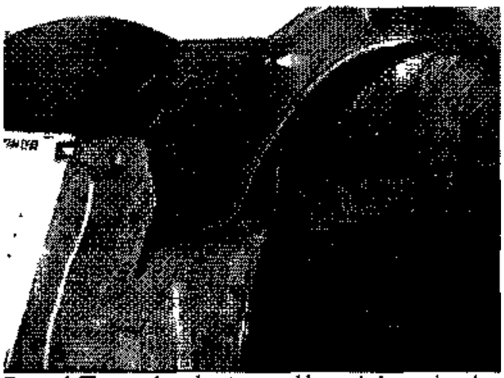


Figure 4. The one-drum hoisting machine with the combined liner.

Mining trials of the liner were earned out. These trials demonstrated that dynamic loads had decreased considerably. The wear of the liner is insignificant. The noise characteristics of the machine operation decreased. All this was achieved due to the elastic relaxation properties of the liner, and the selection and calculation of the optimal dimensions of the liner elements.

4 CONCLUSIONS

The conducted trials show that not only the physical and mechanical properties, but also the geometrical characteristics of the liner of the drum of the hoisting machine have an important effect on the durability of the rope and the liner. The use of the combined liner enables us to increase their durability and to considerably decrease the dynamic and noise characteristics of the machine operation.

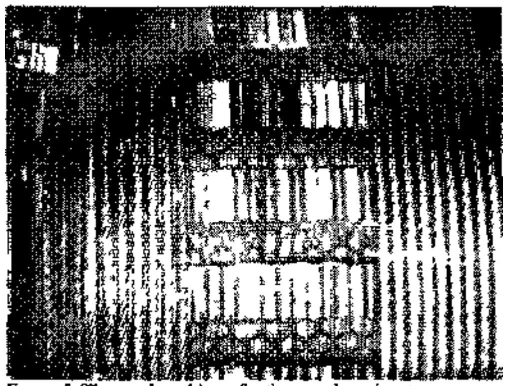


Figure 5. The combined liner for the one-drum hoisting machine

REFERENCES

Franchuk V P & V.V.Franchuk 2000 Determination of tractive capacity of the drum of a multi-rope hoisting machine with due account of flexibility of its elastic liner *Proceeding of the 9th International Symposium on Mine Planning and Equipment Selection (MPES 2000), Athens, Greece, November 6-9, A A Balkema, pp 559-562*

Novikovetal 1978 Theoretical prerequisites of the process of motion transfer of mining transportation machines. In *Problems of dynamics and strength of mining transportation machines* 108-122. Kyiv. Naukova dumka

17th international Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 395-417-4

Technique of No-Pillar Production and Repeated Development of Mineral Deposits with Ice-Rock Laying of Worked-Out Space

N.Zhalgassuly, M.T.Toktamysov, V.I.Galits, B.M.Mukushev & Z.U.Ergusaev
D.Â. Kunayev Institute of Mining, Ministry of Power, Industry and Trade. Almaty, Republic of Kazakhstan

ABSTRACT: This report describes the basic technique, technical opportunities and economic benefits of ice-rock laying of worked-out space in the process of no-pillar production and repeated development of mineral deposits, and also outlines areas for the efficient use of this technique.

1 INTRODUCTION

The aim of the technique is the extraction of useful minerals from seams, deposits or any ore body without leaving pillars of minerals and with complete or partial backfill of the worked-out space or with a structure of artificial pillars composed of ice materials, for example, ice rocks. The technique of *ice-rock* backfill or a structure of pillars is applied with repeated development of seams or ore bodies. Both natural and artificial cold obtained by various methods is applied to ice-rock stowing operations.

The application of ice for stowing is based on the following properties: at a pressure of more than 2 atm. (0.2 MPa), the ice passes from a fragile state to a state that is plastic and viscous. At an environmental temperature of more than 0°C, the temperature of a glacier constantly corresponds to the melting point of ice, 0°C. The melting of ice or freezing of water automatically maintains this state on the surface of the glacier. İt İs calculated that a glacier of 3-m thickness near a daylight area thaws in 300 years, and at a depth of 600 m thaws {due to the heat of adjoining rocks, engineering and ventilation) in 14.2 years.

By laying rocks on ice, and by using cooling agents (for example, solid carbonic acid) and heat insulation, it is sometimes possible to increase the safety of ice at any depth of mining operations.

Ice-rock backfill melts due to convection transfer of cold from the backfill: by thermal flows of massif of containing rocks, from the process apparatus and system of ventilation of wells and ore mines.

The average significant heat provided by adjoining rocks, without geothermal gradient, is 362 kcal/m^3 annually, which causes the melting of a 5-mm layer of ice on the surface of contact with adjacent strata.

As energy equal to 1 kW=3.6 M/ causes the melting of 10.75 kg of ice, the energy provided by the apparatus (5 kW/t) melts 53.75 kg or 0.06 m^3 of ice per ton of extracted ore, or 0.18-0.24 m of ice per 1 m of extracted ore.

Convection transfer of cold and melting of the backfill due to ventilation of the mine workings round the backfill are determined by known methods of heat mass transfer. The quantity of energy required for the formation of ice-rock backfill depends on the method of cooling agent reception, the method of obtaining the ice and the backfill technique, and İs within the range of 5-15 kWh per 1 t of extracted ore. As the cost of electrical power is 0.035 US dollars per kWh, the cost of backfill is 0.175-0.525 dollars/t of extracted ore, which İs much less than the cost of traditional backfill.

2 CONCLUSION

The benefits of this technique are reduction in cost of the stowing material and stowing works, and an increase in profit in mineral production of approximately 30 %. With the use of ice-rock backfill, there is the opportunity for the smooth lowering of overlying rocks in goaf on large areas under protected objects without significant damage. Thus, the ecological situation in wells, ore mines and on the surface is improved due to the use of ecologically pure stowing materials - water and cooling agents.

The technique can be applied with any mining methods which are now applied with goaf stowing with other materials and where the development of minerals is carried out under protected structures.

17th International Mining Congress and Exhibition of Turkey- IMCET2001, © 2001, ISBN 975-395-417-4

Çayeli Underground Cu-Zn Mine

M.Yumlu
Çayeli Bakır İşletmeleri AŞ (CBI), Rize, Turkey

ABSTRACT: Çayeli Bakır işletmeleri AŞ (CBI) is a joint venture between Inmet Mining of Canada (49%) and two Turkish partners. Eti Holding AŞ (45%) and Gama Endüstri AŞ (6%). During construction and through the first five years of production, the operator, Inmet Mining, has successfully brought a 2880-ton-per-day underground Cu-Zn mine into production. The Çayeli underground mine is located at Madenköy, about eight kilometers south of the coastal town of Çayeli on the northeast Black Sea coast of Turkey. In 1994, CBI began development and production from a massive sulfide ore body dipping to the northwest at about 70 degrees. A retreat transverse/longitudinal long hole stoping method with post backfill was developed utilizing trackless mining equipment. CBI has been able to achieve low mining costs and high productivity using this mining system. This paper will briefly review the Çayeli mine in general. Specifically, a brief history of the mine, the geology of the deposit, the mining method, backfilling, ground support, ventilation and dewatenng, ore haulage, hoisting and finally staffing will be outlined.

1 INTRODUCTION

Çayeli mine is located in the Black Sea region of northeast Turkey. The mine is approximately 28 km east of Rize and 100 km west of the border with Georgia (Fig. 1). The nearest town is Çayeli, 8 km away. The Çayeli mine site is located in the foothills of the Pontid mountain range. This region is characterised by high topographic relief and high rainfall.

The mine is operated by Çayeli Bakır işletmeleri AŞ (CBI) and produces copper and zinc concentrates by processing massive sulphide ore.

The concentrates are transported by truck to the port of Rize and are then shipped to domestic and overseas smelters.

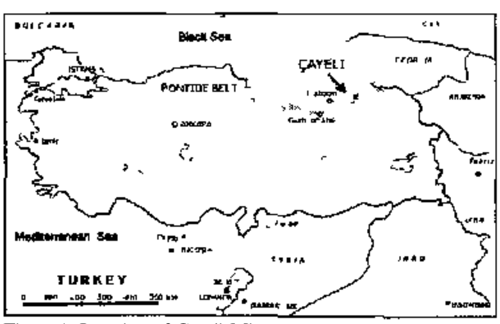


Figure 1. Location of Çayeli Mine.

CBI has been in operation since August 1994 and has undergone a series of technological changes since then, such as the completion of a shaft to replace truck haulage from underground, and the introduction of paste backfill. Ore production has increased virtually every year, and the 2001 target is 1.0 million tonnes of ore

This paper gives a brief account of the discovery of the Cayeh ore deposit, the background of CBI. the on-site operations and the outlook for the future.

2 DISCOVERY AND BACKGROUND

Mining activities along the Black Sea coast and at Çayeli date back at least a thousand years. At the turn of this century, minor exploration by the Russians was reported and, between 1930 and 1955, various shafts and adtts were driven and some minor production took place.

The work which led to the present mine was started İn 1967 by the Turkish Mineral Research and Exploration Institute (MTA). MTA carried out a geophysical *survey* and drilling programme, and drove an adit into the massive sulphide ore. In 1981, CBI was formed as a joint venture between Etibank (now Eti Holding AS), Phelps Dodge and Gama Endüstri AŞ to develop the ore body. Phelps Dodge sold its share to Metall Mining (now Inmet Mining) in 1988. Further underground work and metallurgical testing were done in the period 1988 to 1991. After positive results, a production decision was made and site construction started in 1992. The basic mine infrastructure was commenced in 1993 and in August 1994 the first concentrate was produced. Table 1 below presents a summary of the operation.

The mining rate has increased, from 1350 tonnes per day in 1995 to 2880 m 2000 (Table 2). Recoveries and concentrate quality have improved since start-up. The total capital cost of the project was over $US 200 million.

Table 1 Summary of the operation

1967	First geophysical survey and drilling program by MTA
1981	CBI was formed as a joint venture between Etibank (now Eti Holding AS). Phelps Dodge and Gama Endüstri AŞ
1988	Phelps Dodge sold its share to Metal! Mining (now Inmet).
1988-91	U/G work and metallurgical testing was done
1992	Site construction started
1993	Basic mine infrastructure commenced
1994	First concentrale was produced in August
1995	Engineering and design for shaft started
1998	Shaft commissioned in July
1999	Paste backfill plant commissioned
2000	Achieved 4 million tonnes production since start-up and mining rate increased from 1350 t/dml995to2880t/d.

Table 2. Main operational statistics

Year	Production, Tonnes	Feed grade	
		Cu%	Zn%
1994	70,641	4 48	7.72
1995	485,815	3.66	7 55
1996	654,448	3.53	8 54
1997	761,608	4.71	704
1998	707,992	4.56	6.63
1999	896,749	511	5 34
2000	860,763	4.87	4.45
Total	4,438,016	4.51	6 42

3 GEOLOGY, ORE RESERVES AND EXPLORATION

3.1 *Ore reserves and exploration*

The January I[s1], 2001 estimate of the resource (measured, indicated and inferred) was 15.9 million tonnes with 4.13% copper, 6.18% zinc, 0.70 g/t gold and 53 g/t silver at a 2.5% copper equivalent cut-off. Mining commenced in 1994 and up to the end of 2000, 4.44 million tonnes with 4.51% copper and 6.42% zinc were mined (Table 2).

The continuing exploration program has led to the indication and identification of additional geological resources adjacent to the main ore body and below the present lowest level, which add up to 5 million tonnes of mineable reserves beyond the original ore resources reported in 1994.

Two recent exploration successes are the far north extension of the Main Ore Zone and the down-dip extent of the Deep Ore Zone. The current focus of exploration is the Deep Ore, which is still open along the strike and down dip, where the next phase of production is planned.

3.2 *Regional and heal geology*

The eastern Black Sea volcamc province (Pontid Belt) bounding the eastern Black Sea coast of Turkey extends for over 500 km from the west of Samsun to the Lesser Caucasus in the Republic of Georgia. It is cut off to the south by the North Anatolian Fault. The Pontid Belt is thought to be part of a large ensialic island arc system of Jurassic to Miocene in age, comprised of calc-alkaline and tholeitic volcanic rocks and flysch-type sediments. During the Mid- to Late Cretaceous period, the Pontid region was dominated by mafic arc volcanism that evolved to explosive felsic volcanism. The Çayeli deposit was formed during the transition from mafic to felsic volcanism.

The Çayeli ore body occurs at the top of a felsic flow/dome sequence. The overlying sequence consists of interlayered tuffs and basalts. The deposit dips to the northwest at approximately 70° and stratigraphie tops are to the northwest.

The ore body does not outcrop but strikes parallel to a small valley where there are extensive exposures of highly altered material which forms the immediate footwall of the massive sulphides.

The ore body has been divided into two parts, the upper Main Zone and lower Deep Zone. The two zones are separated by a major discontinuity, which is interpreted to be a synvolcanic growth fault. This feature strikes sub-parallel to the ore body and dips to the east. The two zones are in contact at the southern end of the deposit and become progressively more widely separated towards the north.

3.3 *Description of the deposit and ore types*

The Çayeli deposit is a volcanogenic massive sulphide with many affinities to bodies of the Kuroko type. Mineralisation is known over a strike length of 920 metres. The measured resource has a strike length of about 600 metres, vertical depth of over 400 metres and varies in thickness from a few metres to 80 metres, with a mean of about 20 metres. The hangingwall sequence is a series of intercalated acid to intermediate pyroclastic and basaltic layers with some minor carbonate beds. The footwall consists of altered to highly altered acid volcanics and pyroclastics.

The Main Ore Zone consists of massive sulphide conglomerates, breccias and sandstones with more than 90% sulphide minerals and minor gangue. The major sulphides in the massive sulphide are pyrite, chalcopyrite and sphalerite with minor galena, bornite, and tetrahedrite. Gangue minerals include barite, dolomite, quartz, sericite and kaolinite.

Immediately underlying the massive sulphide ore İs a zone of felsic volcanics which have been altered to clay. Sulphides within this zone consist of disseminated pyrite and veins of pyrite and chalcopyrite.

A sulphide stockwork zone underlies the thickest sections of massive sulphides. It consists of siliceous multidirectional veins which are mineralised with pyrite and chalcopyrite.

The Main Ore Zone consists of two overlapping convex lenses with a strike length of 450 meters and a down dip extent of approximately 200 meters. The Deep Zone occurs below the structural discontinuity that cuts off the Main Ore Zone in a down-dip direction. The ore lithologies are the same as in the Main Ore Zone, but there is a higher proportion of clastic and black ore and proportionately less yellow ore. The ore stratigraphy is not as well defined as in the Main Ore Zone. There is limited information about the Deep Ore Zone, but it appears to be a sheet-like zone with variable thickness.

There are four main ore types. Stockwork ore underlies the main massive sulphide lens and is thickest where it is in close proximity to the synvolcanic fault which offsets the Main and Deep Ore lenses. Yellow ore (copper-rich) occurs immediately above the stockwork zone. Black ore (zinc-rich) occurs above or lateral to the yellow ore. Clastic ore, which is characterised by sphalerite fragments, occurs at the top or around the edges of the ore body.

The stockwork ore is characterised by veins of pyrite and chalcopyrite which occur in the footwall rhyolite. The mineralogy *of this* ore type is simple, with relatively coarse chalcopyrite and pyrite.

The yellow ore consists of pyrite and chalcopyrite clasts, up to 20 cm in size, in a sulphide matrix containing less than 10% sphalerite. The mineralogy of this ore type is also simple.

The black ore consists of pyrite and chalcopyrite clasts from 2mm to >20 cm in diameter in a fine-grained matrix of pyrite and sphalerite containing more than 10% sphalerite. Locally, the sphalerite contains fine inclusions of chalcopyrite, which affects the quality of the copper concentrate.

The clastic ore consists of pyrite, chalcopyrite and sphalerite clasts, from 2 mm to more than 20 cm in diameter İn a sulphide matrix. Sedimentary textures, most notably graded bedding, are present. The sphalerite clasts are diagnostic of this ore type. This ore type is the most metallurgically difficult ore type owing to the fine inclusions of chalcopyrite in the sphalerite - the so-called chalcopyrite disease.

4 MINING

4.1 *Mining Method*

The mining method employed at CBI has been designed for 100% extraction with complete pillar recovery, while allowing no perceptible surface subsidence. The mining method is retreat transverse and/or longitudinal long hole stoping with post backfill.

The ore body *is* accessed from a ramp system located in the hangingwall (HW) and a production shaft located in the footwall (FW) of the ore body. The main levels are at 80-100-m vertical intervals with sublevels at 20-m intervals (Fig. 2).

The ore is developed by driving strike access drifts with a cross-section of 25 m^2 along the hangingwall or footwall or in the center of the ore body to the boundaries. Slope preparation is carried out by driving sill drifts across the strike to the hangingwall or footwall, or in the case of low-grade areas, to the boundary of the economic cut-off grade (Fig. 3a). The sill drifts are 7 m wide by 5 m high. The length of the sill drifts depends on the thickness of the ore body and the location of the strike access drifts. The average length of the sill drifts/stopes is presently 35 m.

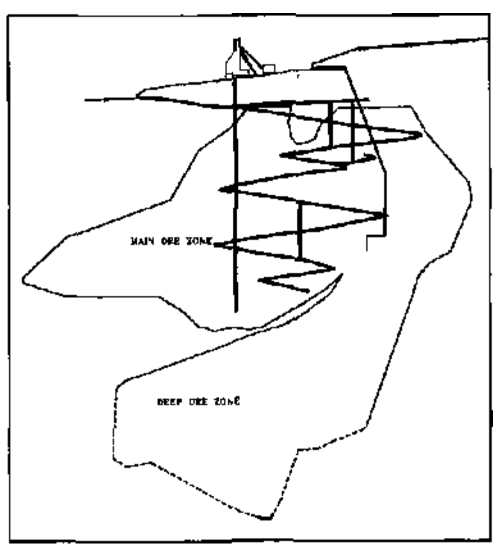


Figure 2. Schematic view of mine infrastructure.

Slope production comprises the extraction of the 15-m-higb bench between two sill drifts (Fig. 3b). A drop raise is excavated between sublevels at the end of the sill drifts. The raise is widened out to a slot to create a free breaking surface. The remainder of the bench is blasted towards the open slot (Fig. 3c). The ore is mucked from the lower sill drift using remote-controlled LHDs. The open stope is then backfilled to the floor level of the upper sill drift (Fig. 3d). The backfilled floor becomes the mucking floor for the next lift. Once two adjacent primary stopes are completely backfilled, the intermediate primary pillar can be mined as a secondary stope. The primary stopes then become backfill pillars. The secondary stopes are also backfilled. The overall sequencing of the mining method is as follows; *(i)* retreat from the boundary of the ore body to the central pillar, *(u)* retreat up dip from main levels, *(lii)* alternating primary and secondary transverse stopes, *(iv)* secondary stopes are mined between primary stopes after consolidation of the cemented backfill, and *(v)* completion of a mining area (main levels) by mining tertiary stopes (longitudinal sloping) in the strike direction between strike access drifts.

4 2 *Drilling and Blasting*

For drilling ore and waste drifts Atlas Copco 282 two-boom electro-hydraulic jumbos with a drill rod length of 4.0 m are used (Fig. 3a). The drill hole diameter İs 45-48 mm. The advance per round is 3.6 m.

In stoping, the initial slot between two levels is prepared by a Cubex Megamatic drill machine fitted with a V-30 Machines Roger raise drill (Fig. 3b). The Cubex drills a hole of 203 mm from the upper to the lower level or vice versa, then enlarges this with a 762-mm reamer head from the bottom up. Stope blast holes are drilled with Tamrock H695 Solomatic top hammer drills (Fig. 3b). Blast holes with a diameter of 64-89 mm are drilled in a variety of patterns, depending on the ore and stope types. The blast holes can be drilled either from the upper or lower sill drift. Stopes are usually completely drilled before production starts.

The main blasting agent is ANFO with a NONEL initiation system, which is transported by Paus trucks and pneumatically loaded. Blasting of the stopes is carried out sequentially with mucking, blasting one to two rows at a time until the stope is completed (Fig. 3c). There İs a central blasting system in use and blasting times are at the end of shifts.

4.3 *Mucking and Haulage*

The ore from development faces and stopes is mucked by Wagner LHDs equipped with ejector buckets. Two sizes of LHD are İn operation; for development, ST 6C is used, and for stoping, ST 8B. All the ST 8Bs and one ST 6C are equipped with remote control. Remote control operation is mandatory in stope production The ore is loaded onto Wagner MT 400 series mine trucks with a nominal capacity of 28 tonnes.

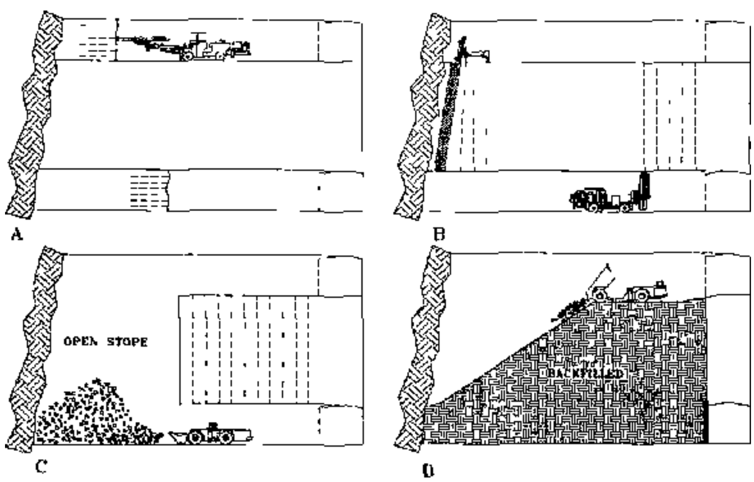


Figure 3. Mining method and mining cycle, a) development, b) drilling, c) blasting and mucking, d) backfilling

Until the middle of 1998 (before commissioning of the shaft), all ore was hauled via the ramp directly to the ore storage bins on the surface. The trucks now dump the ore at the ore pass loading stations on levels 960, 940 and 920. The haulage distances on levels 900 to 960 do not exceed 200 meters. Ore from the upper levels is hauled to the 960 level.

The two ore pass systems allow the transport of two different ore types to be controlled. Generally, the spec ore consists of two ore types, yellow ore and black ore, which are produced simultaneously. The clastic ore, which currently accounts for about 15% of the total ore mined, is hauled in campaigns, replacing one of the other ore types.

The ore passes are of 400 tonnes capacity between sublevels and must be pulled constantly. Ore pass hang-ups rarely occur due to the excellent fragmentation of the generally friable ore at CBI.

On the 900 level, the ore is loaded by an ST 6C from the discharges of the ore passes to a feed hopper with a grizzly, which limits the block size to 300 x 300 x 450 mm. The amount of oversized material from the ore passes is fairly limited.

The haulage distance from the ore passes to the feed hopper is 25 m and 50 m, respectively. The different ore types from the ore passes are strictly separated by feeding only ore from one ore pass at a time. When the ore type changes, the feed hopper is emptied completely before the feeding of another ore type.

Waste rock is hauled from the development faces by trucks either to the surface or directly to stopes for backfilling.

4.4 *Backfilling*

Backfilling is an essential part of the mining process at CBI (Fig. 3d). Several types of fill are used for backfilling, namely, cemented rockfill (CRF), cemented paste fill (PF), and uncemented waste fill (WF).

For primary stopes, CRF and/or PF are used. The primary requirement here is to stabilize the ground to permit the extraction of the adjacent ore. The fill is exposed by the subsequent ore extraction, and hence must have sufficient strength to support itself when the constraining ore walls are removed. The primary stopes are filled with CRF or PF with 5% cement depending on the availability and stope requirements.

Secondary stopes, except for the brow section, which will be exposed during tertiary extraction are backfilled with uncemented development waste material or 2% cemented paste fill. Here, the voids generated after secondary extraction are filled to provide regional support. Use of WF for the most part facilitates disposal of waste from development mining, eliminating the need to truck or skip the material to surface waste stockpiles. A portion of secondary stope fill will be exposed during extraction of tertiary stopes. In order to minimize dilution, secondary stopes on the access drift side are filled with CRF or PF (with 5% cement).

Tertiary and longitudinal stopes are backfilled with cemented rock fill and/or paste fill with 5% cement.

The uppermost sill drifts on main levels usually require the backfill to be tight against the back to minimize spans and to improve regional stability.

Generally, cement dosing for CRF and PF is 5%. However, the stope geometry and the prevailing mining conditions dictate the final dosage of cement in backfill.

4.5 *Ventilation*

Fresh air for primary ventilation to the mine is supplied through two down cast raises and the shaft, at a combined maximum rate of 245 m/s. The ventilation raises are equipped with one 220-kW fan each, with a maximum capacity of 110 mVs. The two ventilation raises are located in the hangingwall, with a diameter of 3.6 m each and inclination of 70 degrees. On the surface, these raises are accessed by a horizontal drift. Ventilation raise No. 1 (In section N1730), serving the mining area between levels 900 and 980, reaches down to the 960 level, and then continues in section N1710 to the 920 level. Ventilation raise No. 2, located in section N1710, ends at the 1000 level and serves mining areas above the 1000 level only. The shaft is equipped with a 30-kW fan, which provides 25 m^3/s fresh air through a 900 mm ventilation duct to the shaft sump at the 830 level.

The fresh air from the raises is diverted on intake air levels to the active mining areas. Several raises within the ore body, with diameters between 0.8 and 3.0 m, serve as exhaust raises. Exhaust air is returned through exhaust levels (typically the top level of a mining area) to the main ramp and to the exploration ramp (above the 1000 level only).

Table 3. Present ventilation system.

Ventilation raise No. I intake	80 m^3/s
Ventilation raise No. 2 intake	80 m^3/s
Shaft intake	25 m^3/s
Hangingwall ramp exhaust	160 m''s
Exploration ramp exhaust	25 mVs

Presently, 22 auxiliary fans with a total installed power of 957.5 kW support the diversion of the airflow to meet actual requirements. Approximate parameters of the present ventilation set-up are presented m Table 3 below.

The total ventilation requirement of the mine is calculated using the total kW of underground machinery in use in accordance with Ontario, Canada regulations (0.06 ra^3/s per kW). At CBI, Cogema fans are used due to their low noise generation, The ventilation network of the mine is regularly measured and calculated using VNETPC software.

4.6 *Ground Support*

The hangingwall and footwall are normally weaker than the ore body and require additional support. The ground support measures applied are adapted to the ground conditions and to the expected operation time of drifts. The ground support measures consist of: *(i)* Steel Fiber Reinforced Shotcreting (SFRS) or Mesh Reinforced Shotcreting (MRS), *(ti)* standard bolting with mesh and split sets (2.4 m), Swellex (2.4m) or Super Swellex holts (3.3-4.0m), and *(in)* cable bolting (9.0 m).

The equipment employed in ground support is a Normet shotcrete applicator, with shotcrete transported from a batch plant on the surface to underground by 5.0-m^3 Normet and 2.5-m^3 Paus 10 concrete mixers and Memco MacLean platform bolters. The holes for cable bolts are drilled by Tamrock Solomatic drills.

All waste drifts are supported with SFRS or MRS and bolts. The bolt pattern is 9 bolts per row (Super Swellex) at a row distance of 1.5 m. The shotcrete thickness is about 10 cm. SFRS is applied in one pass, while MRS is applied in two passes with a mesh layer in between.

The hangingwall, footwall, and central ore development access drifts are supported by split sets and Swellex bolts. In addition, 9-m-long fully grouted cable bolts are regularly installed in a pattern of 4 bolts per row with a row distance of 2.5 m. where required shotcrete is also applied.

Sill drifts are supported by standard bolts (approx. 90% split sets) İn a pattern of 9 bolts per row with a row distance of 1.25 m.

Considerable bolting and shotcreting is required in the rehabilitation of ore development and sill drifts. Rehabilitation requirements due to deterioration of installed support demand tight sequencing of development and stoping.

4 7 *Mobile Equipment*

The main elements of the current underground mobile equipment fleet are listed in Table 4.

4.8 *Mine Drainage and Dewatermg*

The main dewatermg pump is installed adjacent to the shaft on the 900 level. Water flows through drainage holes and ditches to the 900 level main dirty water sump and is pumped to the surface through the shaft. The main dewatermg pump used is a 132-kW Geho duplex double-acting crankshaft-driven high-pressure piston diaphragm pump, model ZPM 700. The pump operates at 65 mVh at an operating pressure of a maximum 53 bar.

Furthermore, a standby pumping station on the 1000 level pumps clear water to the surface through the exploration ramp, employing two Mather and Piatt pumps.

The total mine water pumped to the surface on average is 25 mVh (approximately 40-45% of which is service water consumed underground).

4.9 *Hoisting and Surface Transport*

Two conveyor belts (100 m and 15 m long) with a feed rate of 350 tonnes/hr connect the discharge of the feed hopper with a 50-m^3 feed bin at the shaft loading area. The two skip loading hoppers can take 5.67 tonnes each. The skips with a volume of 1.89 m^a each operate with an average load of 5.35 tonnes. The rated capacity of the shaft, based on a hoisting speed of 7.7 m per second, is 260 tonnes ore per hour.

On the surface, the skips can discharge into two bins, a 200-tonne bin assigned for ore and a 50-tonne bin assigned for waste. The waste ore bin can also take ore The ability to use the waste ore bin for ore is important in avoiding delay while the ore type changes and the 200-tonne ore bin is not empty. The waste bin then acts as an intermediate buffer.

From the shaft, the ore is transported in Volvo A35 trucks with a payload of 32 tonnes to the ore stockpiles. The transport distance is approximately 400 m.

Table 4 CBI's primary mobile equipment

Jumbos	Three Atlas Copco 282 twin-boom face Jumbos
Production drills	Two Tamrock H629 Soiomatic drills One Cubex Megamatic ITH drill machine fitted wim V-30 Machines Roger raise drill
Bolters	Two Memco MacLean platform bolters
LHDs	Three Wagner ST8B Scooptrams Three Wagner ST6C Scooptrams One JS 200
Trucks	Four Wagner MT 433s (30 t) Two Wagner MT 4336Bs (33 t)
Shotcreling machines	Two Normet Unimixers (5m^1) One Normet Spraymec 6050 WPC (Second Normet Spraymec on order) One Paus mixer
Utility vehicles	Three Fargo scissors-lift trucks Five Paus platforms Two Paus ANFO trucks One personnel earner

4.10 *Supplies*

Shotcrete is prepared at a surface batch plant located close to the ramp portal and transported in mixer trucks of 2.5 and 5.0 m^3 capacity to the underground working places.

Consumable material supplies are transported underground by Paus truck.

The electrical energy supply to the mine is provided by 6.3-kV feeder lines through the shaft and the main ramp to 300- or 500-kVA substations on different sublevels. Distribution from the substations is at 380 volts.

The diesel fuel station is located near the ramp portal on the surface. LHDs, trucks and transport vehicles refuel at this station.

Compressed air supply is provided by four compressors (each 160 kW,10 bar, 380 lt./sec) on the surface. The main feed pipes, 100-200 mm in diameter, are installed in the shaft, in one ventilation raise, and in the main ramp.

Water supply lines, 50-100 mm in diameter, are installed in the shaft, in one ventilation raise and in the main ramp.

5 ACCESS AND INFRASTRUCTURE

5.1 *Ramps*

The main access to the mine for mobile equipment is provided by the ramp, located 60-100 m into the hangingwall of the Main Ore Zone (Fig.2). With the portal at the 1096-m level and with a cross-section of 25 m^2, the main ramp currently extends at an inclination of 15% down to the 840 level.

Furthermore, the ramp provides access to the top and bottom of the aggregate backfill raises, serves for equipment, material and employee transport into the mine, served until the middle of 1998 as a transportation ramp for ore truck haulage to the surface, and serves as an exhaust way for ventilation.

Access is also provided via the exploration ramp with a 9-m^2 cross-sectional area, at a 17% gradient down to the 1000 level. This ramp is used as a return airway and for services such as the pastefill reticulation system.

5.2 *Shaft and Ore Pass System*

The mine is serviced by a 5.5-m-diameter, 275-m-deep, vertical, circular and concrete-lined hoisting and man-riding shaft situated in the footwall.

The shaft collar elevation is at 1110 m with the shaft bottom reaching the 830 m level. Underground access to the shaft is at the 900 level only. Access to the shaft on the 900 level is by a 120-m-long cross cut. The shaft is equipped with two skips and a man cage.

The shaft is equipped with a 120"-diameter GEC 863 HP DC, direct drive, single-tooth-type clutch, double drum hoist with two 5.67-tonne-capacity skips. The second hoist will be used to extend the shaft to the 640 level without interrupting production. A small cage is installed for transporting men.

A system of two ore passes with a cross-section of 10.2 m^2 each, located in the central pillar of the Main Ore Zone (section N1760), connects the 960, 940 and 920 levels with the 900 level. The ore passes are concrete lined and provide a storage capacity of 400 tonnes each between two levels.

5.3 *Buildings and site works*

The surface buildings and site works include surface workshops, a warehouse with adjacent fenced yard, mill, shaft, covered-surface stockpile, laboratory, change houses, main administration office, batch plant, pastefill plant, medical building, weigh bridge, and gate house. Sewage reticulation, drainage, landscaping and lighting is also provided.

5.4 *Other Major Installations*

Two backfill raises for aggregate storage are located in the hangingwall. The raises are 3.1m in diameter and are inclined at 85 degrees. The loading level of both raises is the 1080 level. One raise discharges on the 1040 level, the second on the 1020 level.

An underground workshop with two work bays is located on the 1020-m level adjacent to the backfill raise load out. The workshop comprises 100 m of 25-m^2 drift, hosting two equipment bays and a small storage area for parts and consumables.

The explosives magazine, comprising 120 m of drift, has a 40-tonne capacity and is located on the 1040 level.

Refuge stations are provided throughout the mine. The communication system comprises telephones on each level. Crews are also equipped with wireless communication units through a leaky feeder system. Furthermore, mere is a leaky feeder video transmission, which is used for the control of filling operations when backfilling stopes with paste.

5.5 *Water Supply*

Fresh water is supplied to the mine from 7 wells on the bank of the Büyük Dere river. The wells have a supply capacity of 450 m^3/h. Potable water is supplied by the Madenli municipality. The storage facilities for process water and potable water have a capacity of 1000 m^3 and 60 m^3, respectively. The total water requirement of the mine and the mill is estimated to be over 350 m^3/h.

5.6 *Electric Power Supply*

The mine is connected to the national power grid by a single 31.5-kV, 30-MVA-rated overhead power line to the TEK substation north of the town of Madenli. The substation at Madenli is equipped with one 25-MVA transformer, and one 10-MVA transformer as a back-up reserve.

At the mine site, the power line terminates at a transformer and switchyard, where a 31.5-kV/6.3-kV substation equipped with two AEG 15 MVA transformers is installed.

6 MILL AND METALLURGY

6.1 *Milling*

CBI has an onsite concentrator, which performs crushing, grinding, differential flotation and pressure filtration. The concentrator utilizes a conventional differential flotation to produce Cu and Zn concentrates. There are two circuits; a zinc circuit and a copper circuit. Mill feed is prepared by blending various types of ores according to their copper and zinc grades. Crushed ore is fed to the grinding circuit, which consists of two-stage ball milling. The grinding circuit produces a flotation feed of 70% passing 36|im. The metallurgically difficult clastic ore is milled in campaigns. Start-up recoveries and concentrate grades have improved. This was made possible through a series of on-going technological changes.

6.2 *Concentrate Transport and Port Facilities*

The copper and zinc concentrates are transported 30 km by truck to the Rize port facilities. At the port, the concentrate is discharged into a covered stockpile area with a capacity of 30,000 tonnes of concentrate. Reclaim for ship loading is by conveyors. The concentrates are shipped to a variety of smelters around the world.

6.3 *Tailings Disposal*

Full plant tailings are used as a paste backfill which is pumped underground or pumped via an 8-km-long HOPE overland pipeline to an undersea disposal site, which is located 3 km off shore at a depth of 385 m, well below the oxygenated surface waters of the Black Sea.

7 MANPOWER

As of the end of December 2000, the labour force is as given below (Table 5). The majority of workers are recruited locally. Underground, plant and maintenance workers work 7.5-hour shifts, with 3 shifts per day, 7 days per week. The shifts start at 8 a.m., 4 p.m. and midnight. Administrative and support staff work a nine-hour day, five days a week.

Table 5. Work force.

Category	Number
Executive	2
Human Resources	2
Ankara Office	10
Finance	32
Security	15
Mill	52
Mine	127
Maintenance	70
Technical	S2
Safety, Health and Environment	7
Administration	18
TOTAL	387

8 CONCLUSIONS

The Çayeli mine is an efficient low-cost copper and zinc producer. Since start-up the operation has been very successful and has undergone major technological changes. The results have been particularly noteworthy considering much of the workforce had no previous mining experience. The mine has sufficient reserves to last for at least 13 years and there is excellent exploration potential for additional reserves at depth and along the strike.

CBI is confident that the company's current healthy position on the world copper production cost curve can be maintained or improved through better use of manpower and technology.

ACKNOWLEDGEMENT

The author wishes to thank the management of CBI for their permission and encouragement to present this work. Thanks are also due to the engineering staff who contributed to this work.

17th International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

A Mine Site Laboratory from Exploration to Closure: A Case Study

H.Babacan
Çayeli Bakır İşletmeleri, Çayeli, Rize. Turkey

ABSTRACT: Chemical analysis is a vital scientific tool for the mining industry. The key role of the assay-analytical laboratory is to produce this information. At some stage, a decision has to be made to establish a laboratory on site, the earlier the better. The Çayeli on-site laboratory performed 734,707 determinations performed on 175,357 samples over 17 years of operation. Over the past 7 years, savings of USS 4.5m have been made over commercial laboratory rates. Analytical requirements and the feasibility of having an on-site laboratory are discussed, and a case study of Çayeli Bakir İşletmeleri A.S. (ÇBİ) on-site laboratory is presented İn this paper.

1 INTRODUCTION

In mining, from exploration to closure, quantitative measurements are essential. At different phases of project development there are different analytical requirements. In the exploration stages, the emphasis is on analysis of ore samples. At the feasibility base line, environmental analysis of air, soil and water will be needed as well as pilot plant concentrate and tailings measurements. During production, routine analysis of samples for grade control, plant operations and shipment are required as well as on-going environmental checks. At closure, and once the mine has shut down, environmental analysis will continue. The data required by the mining industry should be measurement based, scientifically approved and legally acceptable at all times.

Before creating a laboratory design, in order to fulfill all these requirements, certain criteria should be reviewed, such as: the number and size of the samples, the required determinations, acceptable turnaround time, the required minimum level of accuracy of the assay, and the importance of sample security. Capacity for future expansion İs also important.

Procedures for different purposes may vary and the type of laboratory changes accordingly. However, even in the short-term exploration stage (e.g., 2 years, with a capacity of 10 samples, 2 elements, per day), a mine site laboratory development is advisable and economically justifiable. In addition, access to consulting laboratory facilities is essential for checking, and some quality-control reference samples which represent die matrix of the material on site should be prepared and assayed by a number of independent reputable laboratories.

At Çayeli, the assay laboratory was established during the early stage of the feasibility study and it processed 7000 drill core samples (28,000 determinations) over a period of 3 years. It was expanded during the operation of the pilot plant and was further expanded and relocated when the mine came into production. In its present form, it was planned to process approximately 30,000 samples per year. In the Çayeli site laboratory, the cost per sample to analyse one element İs about US$ 3, which is about 15 % ofcommercial rates.

An on-site laboratory also introduces a high level of training and employment for local people in places remote from major cities.

2 MINING INDUSTRY AND ANALYTICAL SERVICES

2.1 *Necessity of data for a mine*

It is essential for any mining project to have realistic data in order to satisfy all financial and social partners and stakeholders and also, nowadays, the international community. During all stages of a mine's life, from exploration to post-mining, *the* close contact with these parties forces mining companies to identify concerns earlier than ever before and to take necessary measures to deal with them.

The analytical results used by the mining company play an important part in decision making and strategy definition of the company in all phases of a project Therefore, those data should be .objective and acceptable at all times. Realistic data can only be obtained by accurate and careful measurements. Correct monitoring, sampling and reliable chemical analysis are vital tools for mining.

2.2 *Analytical services*

Mining companies often perform the sampling and sample reduction themselves, but three options are available in terms of analytical services:

i. a commercial laboratory anywhere in the world,
ii. a contracting laboratory on or near the site,
iii. an on-site laboratory operated by the mine itself.

Before making a selection or a deciding upon a design, the mission of the laboratory should be determined using a checklist similar to the one presented in Table 1.

Table I. Checklist for design criteria for an analytical laboratory facility.

Order	Itena to be examined
1.	Clear definition of the information required
2	The purpose of the information
3.	Size and slate of the material to be sampled
4	Size and state of the gross sample
5.	Number, size and state of representative samples
6	Full chemical composition and physical properties of the gross material.
7	Sampling and sample preparation methods required
8.	Sample security and sample safety.
9	Anatytes to be determined in the final sample.
10	Concentration range of each analyte in the sample.
11.	Degree of accuracy required for each analyte
12	Detection and lower reporting limits.
13	Degree of quality control and assurance (QCQA)
14	Turnaround lime for each sample and analyte.
15.	Sample *archive/document archive requirement.*
16.	Future analytical needs - expansion flexibility.

An analytical procedure for an analyte is not applicable to all types of samples. The type of laboratory changes with the procedures. Once the design of a mining laboratory is settled, it has a major objective and therefore a scope. Any sample outside this scope should not enter that specific laboratory. This is for the security of the sample and also for the security of the samples İn the lab. For example, a laboratory dealing with rich sulphide minerals or sulphide mineral concentrates should not accept a sample for the analysis *of* trace heavy elements. The work outside the scope of this laboratory should be performed by other consulting laboratories.

In order to check the reliability of these laboratories, the mining company should introduce some reference-check samples and quality control - quality assurance measures for every group of work. There are commercially available certified reference samples. Initially these will be used. But these do not have the same matrix as the mine samples so, a mining laboratory must prepare its own internal reference samples from their own ores and carry out an inter-laboratory data correlation with the contribution of at least 10 different reputable consulting laboratories using different analytical techniques. Inter-laboratory correlation work should continue at intervals throughout the life of the project. A mining company, even though it might have its own laboratory, will continue to support commercial laboratories worldwide with a considerable amount of work in order to maintain the reliability of data.

2.3 *Feasibility of on-site laboratory*

Analytical works and their control are very intensive and should be under the strict control of the mining company. Therefore, it is better to have major jobs done on site. Cost, of course, is another concern. A comparison of site laboratory versus commercial laboratory costs during exploration and pilot plant operation is presented in Table 2. For this study, the site laboratory used to compile mis table analyses 10 samples per day for 2 elements using instrumental methods with a very high degree of accuracy and precision though, in fact, it would have a much higher capacity.

Table 2. Site lab versus commercial or contracting laboratory.

Cost Items	Site Lab. *	Contracting or Commercial Lab.
Sample Preparation	Included	USS 4-
Sample Archive	Included	extra charge
Sample Shipment	Included	US$2-
Number of Analytes	2	2
Assay Cost/Sample	USS 6.-	USS 20-40 **
No of Duplicates	All	1/10
No of Replicates	1/ 10	1/20
No of QC Samples	1/10	1 / 10
Cost - Check and QC Samples	Included	USS 20-40
Turnaround Time	<l day	2 - 20 days'**
Work unit	10 samples	10 samples
Total Cost per Sample	US$6-	US$ 24 - 46
LAB. INVESTMENT		
Building (45 sq.m)	USS 25,000	No
Equipment and Lafa Ware	ÜSS 75.000	

* 250 days a year / 6 days a week / two shifts - *2* chemists, *6* technicians.
** Taken from quotations of various laboratories.
*** More Ihan 30 days observed.

Based on the model above, a mine site laboratory is economically and technically feasible and yields an internal rate of return (IRR) of 43%/year and a high net present value (NPV) for an initial investment of US$ 100,000 at an average assay cost of US$ 3 per element (Table 3).

Table 3. Feasibility of a site laboratory.

	Saving by use of Contracting Lab (24 USD / sample)	Saving by use of Commercial Lab (46 USD I sample)
Annual Cost Difference" (Saving)	US$45,000	USS 100.000
IRR (10 years, i = 12%)	43%	100%
NPV	USS 125,000	USS 386,000

* Saving = (Site Lab. - Contracting or Commercial Lab Facilities)

3 SITE LABORATORY OF ÇBİ

3.1 *History of lab development at Çayeli mine site*

The ÇBİ on-site laboratory was initially planned and developed in May 1984 to perform copper and zinc analysis on drill core and rock samples, at a rate of 10 samples per day, within error limits of 10% using an atomic absorption spectrometer (AAS). The staff consisted of one chemist and two technicians in an area of 45 square meters. After a short while, in 1985, as a result of growing demand for analytical services, the lab was extended to an area of 125 square meters to perform Cu, Zn, Pb, Fe, Ag and Cd determinations by AAS with 99% correlation witfi a capacity of 50 samples (i.e., 250 assays) per day. This capacity was achieved by addition of a rented AAS and a second ring mill pulveriser to the capital equipment, with a staff of two chemists and six technicians working in two shifts.

In 1992, during the site development stage, a new 300-square-metre assay laboratory was planned. The existing laboratory equipment was transferred to its final location with the addition of a second AAS, a portable energy dispersive XRay Fluorescence Spectrometer (XRF), a ring mill pulveriser and a jaw crusher in August 1994. The total investment of establishing the site laboratory was approximately US$ 600,000. The capacity of the present laboratory is timely and accurate (at 95% confidence level) analysis of 150 samples of various types per day on a routine basis, with a staff consisting of 4 chemists and 8 technicians working 7 days a week on two shifts.

At present, the laboratory is preparing to add the capacity to cany out classical methods for zinc and copper analysis on concentrate sales samples to the present instrumental methods. It is also researching the techniques for additional tests and analysis of environmental samples related to the closure plan and different ore geology studies. Splits of all samples that have been processed by the laboratory are in the sample archive and are accessible for inspection. Analytical services from outside laboratories are routed through, and the reports are checked by the on-site laboratory.

3.2 *Functions ofÇBİ on-site laboratory*

The main objective of the ÇBİ on-site laboratory is to perform sample preparation and quantitative analysis of: ore, mineral, mine and mill products, shipment samples, tailings, soil, sediments, plant, surface and underground water samples (Table 4).

Table 4. Main scope of ÇBI on-site laboratory.

Elements/ Analytes	LLD*	Lower Report Level	Method
Cu, Zn, Pb, Fe	1 ppm	001 %	Flame AAS
Ag	1 ppm	lOg/T	Flame AAS
Cd	1 ppm	lOg/T	Flame AAS
Au	1 ppm	O.IgH"	Extraction / Flame AAS
Cu Zn	001 %		Wei chemical / Classical

LLD = Lowrr Limit of Delecuon

The sample preparation facility of the lab consists of a heavy-duty drying oven and dryers, and weighing, sample splitting and size reduction equipment - Jones rifflers, a laboratory-type jaw crusher, a cone crusher, a disc mill pulveriser and 3 ring pulverisers, a dust extraction system and a sample storage and sample archive area.

In addition to routine assays of Cu, Zn, Pb, Fe, Ag, Cd and Au, the laboratory can perform tests and analysis of 11 other analytes (CN\ $SO_4^=$, $SO_3^=$, SO_2, COD, pH, hardness, Ca**, Mg^{++}, NH_3, heavy metals). It can also test for ore oxidation, buffer capacity of rock and the quality of material like lime, xanthates and copper sulphate, using various analytical methods.

3.3 *Annual production figures*

In the 17 years since it started work, a total of 734,707 determinations have been performed on 175,357 various samples in the laboratory (Table 5).

The total cost of laboratory operation in die year 2000 was USS 383,500 (Table 6). Compared to commercial laboratory rates, savings of US$ 4.5 million have been achieved over the last 7 years of operation by having an on-site laboratory (Table 7).

Table 5. Production summary of Çayeli laboratory.

Year	No of Samples	No. of Determinations
1984	1,200	1,500
1985	1,700	3,000
1986	3,800	7,000
1987	700	5,000
1988	–	–
1989	2,600	9.000
1990	184	1,042
1991	708	2,108
1992	825	3,500
1993	1,580	6,575
1994	7,716	35,390
1995	22,358	93.973
1996	26,864	107,899
1997	26,862	111.306
1998	22,573	101,102
1999	27,506	123,295
3000	28,181	123,017
Total	175,357	734,707

Table 6. Cost of Çayeli lab operation - Year 2000

Cost Items	USS/year
Operating Salaries	256,000
Operating Material / Services	35,000
Power Consumption	40,000
Depreciation of Construction	20,000
Depreciation of Capital Equipment	20,500
Consulting Lab Costs	12,000
Annual Total Cost (Year 2000)	383,500

Table 7. Savings in last 7 years of Çayeli lab operations (since mill start-up) compared with commercial laboratories.

	USS
Estimated Commercial Lab Charges USS 44 rate per sample (4 elements)	7,130,640
On-site Lab Cost (7 x 383,500)	2,684,500
Saving over 7 years	4,446,140

3.4 *Reliability of analysis*

In order to maintain the reliability of the analysis done by the on-site laboratory, the following quality-control systems are employed in every operation and assay batch in a shift:

i. only certified calibration standards are used,
ii. internal reference samples are added to every batch,
ui. a blank sample is added to every batch,
iv. some samples from the p-evious day are re-read,
v. some settled survey - umpire lot samples are added to every batch,
vi. duplicate analyses at a rate of 10% are performed,
vii. all results are checked with another method,
viii. 5% of samples are re-assayed on another day,
ix. some dummy samples are sent to consulting laboratories,
x. some blind duplicate samples are included in every batch,
xi. certified reference samples are used,
xii. a good record of data and data analysis is maintained.

As a result of this intensive quality-control and assurance program, the assay results of the Çayeli on-site laboratory agree within relative error limits of 1 %, and a good correlation of data with the survey-umpire laboratories is maintained (Figures I-5).

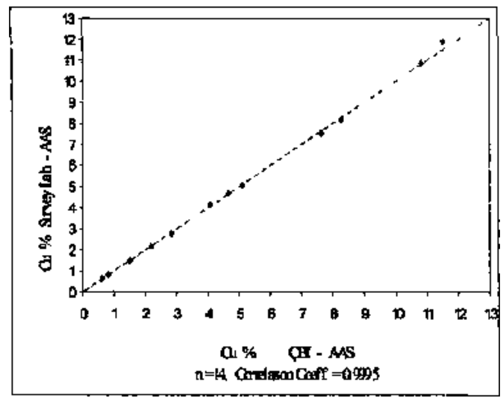


Figure 1. Copper assay correlation on ore samples - ÇBİ site lab versus surveyor laboratory (Nov. 2000).

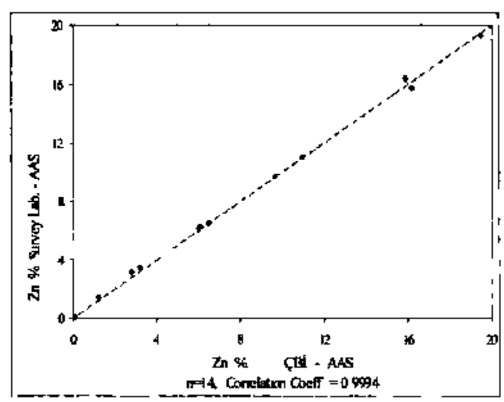
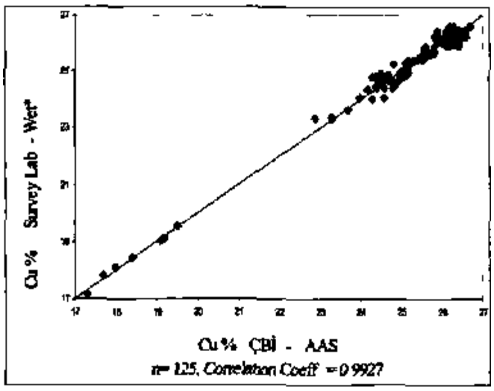


Figure 2. Zinc assay correlation on ore samples - ÇBI site lab versus surveyor laboratory (Nov. 2000).



* Wet = Wei chemical - Classical assay methods

Figure 3 Copper assay correlation on concentrate shipment samples (Year 2000)

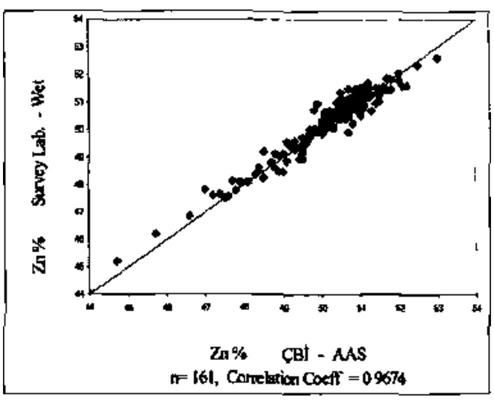
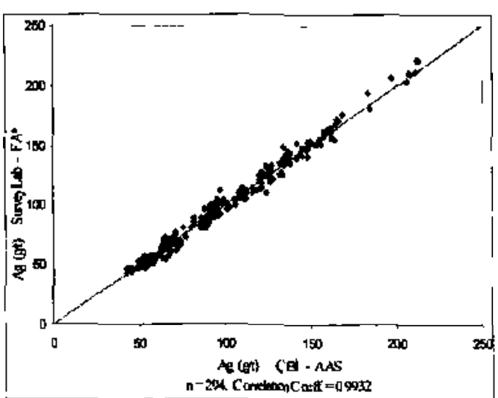


Figure 4 Zinc assay correlation on concentrate shipment samples (Year 2000)



FA = [ırc Assay

Figure 5. Silver assay correlation on concentrate shipment samples (Year 2000)

3.5 *Contribution to the community*

More than 15 workers previously employed in the agricultural sector and 10 chemists have been employed and trained for laboratory work in the 17-year period.

Training in basic chemistry, analytical laboratory work, on-the-job and off-site safety, industrial hygiene and health and environmental issues is a regular daily practice. This high level of training has significantly improved the lifestyle of the employees and their families.

4 CONCLUSIONS

Experience at Çayeli has shown that a mine site analytical laboratory has a very high rate of return on investment and makes a great contribution to the operation and the community. İt İs strongly advised that a laboratory be established early in the life of any mining project, particularly when it is in an area remote from major cities

ACKNOWLEDGEMENT

The author wishes to thank the management of ÇBİ A S. for permission to publish this paper and Mr J B.M Hobbs for assistance in its preparation.

REFERENCES

Garfield, M.F 1997 *Quality assurance principles for analytical laboratories* Gaithersburg AOAC Intl

Lenahan.WC & Murray-Smith, RdeL. 1986. *Assay and analytical practice m the South African mining industry* Johannesburg South African Inst, of Mining and Met.

Hamilton, H M & Yates, EM 1994 Marketing aspects for zinc concentrates. *CIM Bulletin* 983 27-31

Karataş, C 1977. *Endüstri projelerim değerlendirme el kitabı* Ankara. UNDP/UNIDO 77/01 41-49

17th Internationa/ Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-4 7 7-4

Author Index